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ECONOMIC & REGIONAL DEVELOPMENT AGREEMENT

Final Report
TASK 1A - VBM Design Guidelines
Chapter 1 - Cut and Fill Mining

by

Inco Limited, Thompson Division
Thompson, Manitoba

March 1990

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Manitoba

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CANADA - MANITOBA

MINERAL DEVELOPMENT AGREEMENT

RESEARCH & DEVELOPMENT IN MANITOBA MINES

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RESEARCH & DEVELOPMENT IN MANITOBA MINES

VERTICAL BLOCK MINING (VBM) DESIGN GUIDELINES

Chapter 1

Cut and Fill Mining

This chapter has been prepared as part of the

TASK 1
To Develop Vertical Block Mining Design Guidelines
For A Steeply Dipping Orebody;
And To Develop Scaling Techniques
of the
Canada/Manitoba Mineral Development Agreement

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RESEARCH & DEVELOPMENT IN MANITOBA MINES

VERTICAL BLOCK MINING (VBM) DESIGN GUIDELINES

The Vertical Block Mining (VBM) Design Guidelines consists seven chapters, published separately. The seven chapters are:

Summary

1. **Cut and Fill Mining**
2. **Data Collection Systems**
3. **VBM Stope Design**
4. **Drilling and Blasting**
5. **Support Requirements**
6. **VBM Stoping Procedures**

Reference to this chapter should be quoted as follows: "**Vertical Block Mining Design Guidelines Chapter 1 - Cut and Fill Mining**"; Canada/Manitoba Mineral Development Sector B - Mining; DSS File No. 14SQ.23440-4-9147-3; CANMET Project No. 4-9147-3; March 1990

EXECUTIVE SUMMARY

Chapter 1: Cut-and-Fill Mining

The objective of this section of the study is to describe cut-and-fill stoping as it is practiced in Manitoba and to examine the potential for improvements, especially of the procedure used at Thompson in those areas where VBM stoping is not appropriate. Cost and productivities applicable to Manitoba cut-and-fill operations are developed and compared with experience elsewhere.

Two cut-and-fill operations are described: the Thompson Mine and the Trout Lake Mine. These mines represent the application of the cut-and-fill method to narrow and wide orebodies respectively. The Thompson Mine typifies a mature, semi-mechanized operation with limited access to the working place. The Trout Lake Mine typifies an operation which has been developed by ramp access from surface and uses mechanized equipment for all aspects of the operation.

It was found that the two Manitoba cut-and-fill operations demonstrate a satisfactory level of cost and productivity within the constraints of the methods. It is evident that much higher levels of productivity could be achieved by providing ramp access for the method at Thompson and by increasing mechanization, but it is uncertain whether a decrease in cost could be achieved.

The study clearly demonstrates that cut-and-fill remains a viable method, comparable in achievement with VBM in certain geometrical circumstances.

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Chapitre 1: Méthode de minage par coupe et remblai

Ce chapitre a pour objectif de décrire la méthode de minage par coupe et remblai telle qu'utilisée au Manitoba et d'étudier les possibilités d'améliorer de la méthode utilisée à Thompson. Les coûts et la productivité des opérations de coupe et remblai manitobaine ont été déterminés et comparés avec d'autres opérations.

Les deux opérations de coupe et remblai décrites sont celles des mines Thompson et Trout Lake. Ces mines présentent respectivement l'application de la méthode coupe et remblai pour des gisements étroits et larges :

- La mine Thompson présente une opération mature, semi-mécanisée, avec des accès limités au lieu de travail.

- La mine Trout Lake présente une opération qui a été développée à l'aide d'une rampe d'accès en surface et qui utilise des équipements mécanisés pour tous les aspects des opérations.

Les deux opérations coupe et remblai présentent des coûts et une productivité satisfaisants compte tenus des contraintes de cette méthode. Pour les opérations de la mine Thompson, il est évident qu'un niveau de productivité supérieur pourrait être obtenu par le percement d'une rampe d'accès et par une augmentation de la mécanisation. Cependant il n'est pas évident qu'une réduction des coûts pourrait être obtenue.

L'étude démontre clairement que l'utilisation de la méthode demeure toujours attrayante si les résultats sont comparés avec ceux obtenus à l'aide de la méthode VBM dans certaines circonstances géométriques.

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1. INTRODUCTION

The report covers the work carried out in Sub-Task 1 of the main Task 1A. The objective of the sub-task is to describe cut-and- fill and the potential for improvements especially as operated at Thompson. The report presents costs and productivities which are applicable to Manitoba cut-and-fill operations and reviews cut-and-fill elsewhere in the world. Finally, it identifies and quantifies changes which could be made to the conventional system at Thompson.

The report is therefore part of the overall design method requirements for VBM stoping and provides data which can be used for evaluation of the suitability of a particular orebody to cut- and-fill or VBM stoping.

2. CUT-AND-FILL IN MANITOBA

Two cut-and-fill operations are described; the Thompson Mine and the Trout Lake Mine. These mines represent the application of the cut-and-fill method to narrow and wide orebodies respectively.

- Thompson Mine: typifies the more conventional, semi- mechanized operation carried out in narrow orebodies with very limited access to the working place.

- Trout Lake Mine: typifies an operation in a wide orebody which has been developed by ramp access from surface and uses mechanized equipment for all aspects of the operation.

The comparison is more valid than is usual as the mines are both in Manitoba. Employment practices and the operator's expectation of productivity are therefore similar.

2.1 Review of Cut-and-Fill Practice at Thompson Mine

Thompson is located in the Province of Manitoba, 950 miles northwest of the Company's operations in the Sudbury district of Ontario and 400 miles due north of Winnipeg.

The orebody extends over a length of approximately 3 1/2 miles. It lies in a schist zone conformable with the local formation. The dip is fairly steep, about 65 degrees, with local flat-lying areas associated with the structure of the deposit. The ore pinches and swells both along strike and down dip, with further irregularities due to local off-shoots into the wall rocks. Mining widths range from the minimum practicable up to a maximum of around 150 feet. In general, both wall rock and ore are fairly strong.

Thompson Mine was developed in the fifties using the horizontal Cut-and-Fill method in order to achieve maximum extraction of the relatively narrow (av. 5m or 15 ft.) ore. Stoping equipment consisted of airleg drills, stopers and air-operated slusher hoists. Filling was accomplished by hydraulically-placed classified mill tailings.

Since mining began a number of important changes have taken place. These include:
include:

- Levels, originally at 61m (200 foot) intervals, were increased to 122m (400 foot) in 1972 to decrease the amount of waste development and resultant capital expenditures.

- The uncemented fill used initially has been superceded by a cemented fill; usually at a 25:1 ratio.

- The top 10m (30 feet) of the cut-and-fill stope was usually extracted by undercut methods. This undercut portion has increased as mining got deeper.

- Slushers have been replaced by 2 cu. yd. loaders.

Conversion to Mechanized Cut-and-Fill was started in 1967. This consisted of replacing slushers with scooptrams and relocating stope ore passes into the footwall. By 1971, twenty-nine scooptrams were operating in Cut-and-Fill stopes and five scooptrams in Underhand Cut and Fill pillar stopes. In the majority of stopes, the scooptrams were captive, and could remain in the stope for many years.

Where slushers were used, stope development took place at sill heights of up to 10m (30 ft.). With the advent of scooptrams, stope development took place at base of rail.

Over the years considerable effort has been put into increasing productivity and decreasing costs. But, within a system where access to a working area is confined to a small raise, there is a limit to what can be achieved. This prompted investigation into other methods and in 1979 Vertical Block Mining (V.B.M.) was introduced at the Thompson Mine. Trial stoping proved the value of this technique as a safer, more rapid method of mining.

The present status of stoping at Thompson Mine is shown in Table 1.1.

Table 1.1 Stoping Method at Thompson Mine

Stoping Method	Producing Stopes	Proportion of Production %	Stope Production Rate Tonnes/Day
Cut-and-Fill (LHD)	4	14	190
Underhand Cut-and-Fill (LHD)	5	4	43
Vertical Block Mining	20	82	223

The production advantages of VBM are evident from the table above. It is important, however, to identify the potential for improving the present cut-and-fill system or changing to a different, more mechanized, system.

The Cut-and-Fill system at Thompson is illustrated in the following figures:

- Figure 1.1 Isometric View of a Typical Mechanized Cut-and- Fill Stope.

- Figure 1.2 Plan of a Typical Mechanized Cut-and-Fill Stope.

- Figure 1.3 Longitudinal Section of a Typical Mechanized Cut- and-Fill Stope.

The present system can be divided into seven steps.

- Development
- Opening a New Cut
- Regular Mining Advance
- Excavation Support
- Special Procedures
- Timbering
- Sandfilling.

2.1.1 Development

Cut-and-Fill stopes are laid out with strike dimensions, where possible, of approximately 240m (800 feet). Historically, the stopes were silled at an elevation of 10m (30 feet) above the base of rail after the boxholes had been developed; or, at an elevation of 3m (9 feet) above the base of rail ,thereby, mucking through the gangway into cars. In more recent times, development of the stope begins with the driving of a strike drift in the orebody along the footwall contact. This drift is commonly called a sill drift, and is driven at the sill elevation, generally at the base of rail.

A footwall extraction drift or haulage is developed in the footwall rock at a stable distance ,usually 10m (30 ft.) or more, from the stope.

The widening of the ore sill drift to the hangingwall contact is done by slashing where the orebody is less than about 8m (25 ft.) wide. Where the orebody becomes wider than about 8m (25 ft.), additional ground support is achieved by leaving a rib pillar, Fig. 1.2. The back and walls are completely bolted and screened as mining progresses.

Sill timbering consists of building stringers and a log mat for the recovery of the crown pillar of the stope beneath and the construction of the bases for the service manways.

The excavation of boxholes in the extraction drift begins when a minimum of 0.6m (2 feet) of 10:1 cemented fill has been poured over the log mat. Waste can then be dumped in the stope as fill.

Three bored raises for ventilation and services are normally driven at the very beginning of the development of a Mechanized Cut-and-Fill stope.

2.1.2 Opening New Cut

The new cut is opened in an area of the stope suitable to the mining cycle as illustrated in Figure 1.4.

2.1.3. Regular Mining Advance

Regular mining advance is a repetitive cycle, including excavation support, drilling and blasting, advancing services and muck removal as shown in Figure 1.5. The dominating feature of the Thompson cut-and-fill system is the lack of cycle dependence between drilling and blasting and ore loading.

Drilling, blasting and excavation support are done off the muckpile. The muckpile being essential as a working platform. In practice a comparatively long muckpile is carried between the face and the ore loading position. This has the effect of smoothing the influence of LHD non-availability, a significant problem when the LHD is captive.

The productivity per manshift and the time spent on the various mining activities are discussed in section 3.3. It is interesting to note how little time, only 12.3 percent, is spent in drilling and blasting. This indicates the importance of providing alternative faces especially as the LHD utilization is as low as 36% on a single shift basis and 13% for three shifts. Although this importance is recognized, studies have indicated that far less than 10% of a stope life is operated with more than one face available.

- **Excavation Support**

All roof surfaces are fully supported with expanded shell anchor roof bolts and screen. Walls are also screened down to the limit of the muckpile depending on conditions. Nearly all of the bolts are 2.5m (8 foot) in length with a few 4.9m (16 foot)bolts used when conditions are thought to warrant the longer support. Figure 1.6 shows a typical bolting pattern. Such a pattern results in approximately 1.5m per bolt (pattern of 1.1 by 1.4m).

Figure 1.7 shows the limit on unsupported span. Various support patterns can be used dependent on face shape and excavation width as illustrated in Figures 1.8 and 1.9. Steps (hanging breast) in the roof may require support as illustrated in Figure 1.10.

- **Drilling and Blasting**

Drilling is carried out with jacklegs and stopers using 22mm (7/8 inch) steel with 32mm (1 1/4 inch) detachable bits. Nearly all the blasting is with Amex and long delay Nonel detonators with the XT cord detonated electrically. Generally, no primer is used.

A typical round in an orebody of average width may contain 18 holes for an advance of 4.6m (15 foot) wide by 2.7m (9 foot) high and breaking to a length of 2.9m (9.5 feet) with 3m (10 foot) steels. This gives a powder factor of approximately 1.2 kg./m^3 (2.0 lbs/yd^3).

- **Advancing Services**

Services cover the following aspects:

  - catwalk; is hung from the back to provide access during filling operations and also to carry air and water lines where the muckpile has been removed; Figure 1.11 shows typical 'catwalk' construction.

- ventilation; where air supply cannot be readily provided by the ventilation raises and manways then auxiliary fans and tubing is installed.

- **Ore Removal**

Ore is mucked with 2 cu. yd. scooptrams, mostly Wagner 2B, and tipped into the footwall orepass. There are usually two or more orepasses in a stope, giving a mean one way tramming distance of approximately 40m (131 ft.).

The average scooptram utilization is 2.9 hrs. per shift. During this period 79 tonnes (87 tons) of ore are trammed resulting in a productivity of 27 tonnes (30 tons) per hour. At a travelling speed of 4 km/hr. a bucket factor 0.85 and swell factor of 0.65 and assuming 50 min. hours then production would be approximately 32 tonnes per hour. Therefore, the achievement of 27 tonnes is reasonable.

2.1.4 Special Procedures

- Footwall Orepass Advance - Figure 1.12 - On every third cut, the footwall orepass must be extended upwards in preparation for a new access. This is normally done in conjunction with the sandfill preparation of an area, in order that the waste rock excavated may be left in the stope.

- Resin Anchored Bolts - Where mechanically anchored bolts do not provide adequate anchorage, normally due to soft ground, shear zones, etc., resin anchored bolts are used.

- Headcover Construction - Figure 1.13 - Where mining progresses through a raise, a standard headcover is built.

2.1.5 Timbering

When mining has been completed in an area of the stope, various timbering tasks are performed in preparation for sandfill:

- Raising Manways
- Catchpit (Slimecatch) Construction
- Bracing and Sealing
- Catwalk Construction and Piping for Fill
- Stringer and Log Mat Construction
- Barricade and Bulkhead Construction

2.1.6 Sandfilling

The Thompson backfill plant and distribution system is illustrated in Figure 1.14. The plant can deliver at rates between 250 and 400 dry tonnes per hour. The pulp density varies between 55 and 70% but a minimum of 60% is used for VBM stopes which are usually cemented to 25:1. Cut and fill stopes are not cemented but special pours for barricade bottom seal and floors vary from 20 to 6:1 and 15:1 for underhand cut-and-fill. The average grading for the sandfill prior to cement addition is:

- sandfill grading: 12% + 48 Mesh
  70% + 200
  10% + 270
  6% + 325
  2% - 325

The percolation rate is 250 mm/hr.

When all sandfill preparation is completed, the filling of the stopes begins, Figure 1.15. In some Cut-and-Fill areas "high" fill is used to give additional hangingwall support. A 1m (3 ft.) opening is left under the back to give breaking room for the next 3m (9 ft.) high breast. This would then give an opening of 4m (12 feet) on completion of mining of a cut, rather than 6m (20 feet) under normal circumstances. This means, however, that a long muckpile is not possible.

2.2 Review of Cut-and Fill Mining Practice at Trout Lake Mine

The Trout Lake Mine is a good example of a modern trackless, Mechanized Cut-and-Fill operation. Information for the following review was extracted from a mine visit and from mine documents, McIntosh and Crocker (1).

The Trout Lake orebody, discovered in 1976, is located some 5 km northeast of Flin Flon under the waters of Embury Lake. Figure 1.16 shows the orebody shape.

The Trout Lake orebody occurs in a fairly typical volcanogenic environment. The ore is hosted by felsic volcanics, mainly fragmental, overlain by andesites and, further to the hangingwall, argillites. Two types of ore are present, namely massive sulphides and disseminated or stringer type sulphides. There are two main zones known at the present time, each of which is made up of a number of lenses. The north zone has upper and lower lenses of massive sulphide and associated disseminated ore. The south zone is made up of a stacked group of four lenses, two at approximately the same hangingwall horizon with the other two at different horizons to the footwall. Both zones plunge to the southeast at relatively flat angles ranging from 25-40 degrees. Dips vary from 45 to 65 degrees to the northeast. Widths of the lenses vary from 1.5 to 25 metres and strike lengths from 100 to over 200 metres McIntosh and Crocker (1).

A joint venture consisting of Hudson Bay Mining and Smelting Co., Limited, Manitoba Mineral Resources Ltd., Granges Exploration Ltd. and Outokumpu Oy was formed in mid-1980 to develop and mine the property. HBM&S was designated as operator. A technical committee, made up of representatives from each company, provides direction to mine planning and operations.

2.2.1 Mine Development

Trout Lake was developed as a trackless, Mechanized Cut-and-Fill operation.

A single decline measuring 4.2 x 5.5m (14 x 18 feet) was driven from surface down towards the west shore where it turned out under the lake, initially with 30m (98 ft.) of rock cover. At this point, a second heading, on the 80m (265 ft.) below lake surface level, was established. These headings were advanced to the ore horizon where both split to service the north and south ore zones, as illustrated in Fig. 1.16.

A ventilation raise to surface was established at the entrance to the 80m (265 ft.) level. The backfill silos, clear water sump, main pump station and escape raise were also developed in this area.

The 80m (265 ft.) level provides service access to the top of each mining block in the north and south zones for ventilation and fill distribution. Electrical power at 4160 volts is supplied to mine substations through this level.

The haulage decline is located 30m (100 ft.) in the footwall of each ore zone. Ventilation and escape raises were established at the north and south ore extremities. Some ground water, which required grouting, was encountered in the footwall workings.

Underground road beds are made up of rock sized at minus two inches plus three-quarter inches. A surface type road grader maintains the haulageways with a ditch along one side.

- **Stope Development**

The vertical stoping interval has been established at 60m (200 ft.). A typical layout is shown in Figures 1.17 and 1.18.

Stope entry from the haulage ramps is gained centrally along strike at 15m (50 ft.) vertical intervals starting with the stope undercut. Each entry serves three 5m (16 ft.) lifts; one at - 16%, one flat, and one at +16%. Entries are backslashed to provide elevation for the next lift.

Stoping begins by establishing the undercut. In the south zone both the footwall lens and the hangingwall lens are undercut. Two ventilation-escape raises are driven in each zone from the undercut to the level above and 127mm (5 inch) fill holes are drilled down into the undercut. While the undercutting and raising is in progress, a drainage level is established below the stope, leaving a 10m (33 ft.) sill pillar. To provide rapid drainage of the hydraulic fill, drain raises are driven in the footwall and hanging wall lenses at approximately 100m (330 ft.) intervals on strike . Settling sumps and a pump station are established on each drainage level. Water is pumped to the pump station above. The drainage level also provides top access to the stope below.

2.2.2 Mining

From the stope entries, 5m (15 ft.) high breasts are carried the length of each stope. Work is cycled, whenever possible, so that as one end is being mined the other end is being prepared for filling. Where multiple lenses exist they are carried up simultaneously.

Electric-hydraulic jumbos with 4.3m (14 ft.) steel and 44mm (1 3/4 inch) button bits are used for all production and major development. When breasting in a stope, a hole pattern of 1m (3 ft.) burden and 1.8m (6 ft.) spacing is used with holes offset on each line. Backholes are drilled on a 760mm (30 inch) spacing and traced with B line to minimize overbreak. The powder factor is approximately 1.2 kg./m (2.0 lbs./yd).

Powder trucks are used to transport and load explosives. An effort is made to use Anfo whenever possible and the mine targets a 4:1 Anfo to stick powder ratio. Nonel is used for priming and all blasts are initiated electrically from designated locations underground. Mine ventilation is such that blasting is permitted

at any time during a shift.

Stope backs are scaled with a mobile hydraulic scaler as well as by hand. Rock bolting is maintained to within 10-15m (33-49 ft.) of the face by two bolting units which place 2.4m (8 ft.) bolts on a 1.2 x 1.2m (4 x 4 ft.) pattern in the hangingwall and back. These rockbolters have two booms, one with a basket for scaling and placing the bolts while the other carries a pneumatic drill.

Ore is mucked from the face by 5 cu. yd. LHD units into footwall ore passes or into 27 Ton trucks in the stope.

Fill dilution is kept to a minimum by leaving about 1m (3 ft.) of broken ore on the sill when breasting. This ore is recovered as a separate operation when each lift is completed.

All development waste is utilized as stope fill; it is covered with hydraulic fill to within 1.4m (4-5 ft.) of of the stope back.

The drain towers are extended each lift and covered with about 1.1m (3-4 ft.) of fill. When the 3 lifts are completed from a ramp access a free draining timber bulkhead is installed to retain the fill.

- **Backfill System**

The hydraulic fill system was designed to place 200 short tons/hr. (dry) of classified mill tailings or screened sand into the stopes as required. Facilities to add cement to the fill are provided but not used.

Dry fill is backhauled by truck from the Flin Flon plant. The surface dump has a holding capacity of some 80 tons. A series of high pressure nozzles slurry the fill at a rate of about 3 tons/min. The slurry flows through a set of 1" cleaning screens leaving any rock or other oversized material behind which is cleaned periodically with a backhoe. Wet fill is stored underground in two excavated silos each having a working capacity of 1800 short tons.

When placing fill in the stopes, material in the silos is reslurried by Marconaflo units located in the bottom of each silo. These units are made up of a rotating, high pressure nozzle which is hydraulically raised into the fill. A concentrated water spray of 320 USGPM rotates at one revolution per minute, slurrying the settled fill which passes through the unit to a mixing tank. The operator maintains flow and pulp density by adjusting the Marconaflo unit or by adding make-up water. A nuclear density gauge provides continuous readings. The system was designed and operates to place 200 tons of dry fill per hour with an average slurry density of 65% solids by weight.

Fill is pumped from the mixing tank by 6x5 slurry pumps through 6" polyethylene pipe on the 80 m level to the top of the ore zones where it flows by gravity through 5" boreholes to the stopes.

In the stopes fill is distributed with 4" ABS pipe. Dams made of wood or waste rock faced with plastic sheeting are used to control the location of the fill. Drain towers (usually 3 per stope) are situated along strike. These cribbed towers are built of 5 ft. flat ended sprags covered with chicken wire and fabrene. A ladderway is maintained within the cribs.

2.2.3 Equipment and Operations

The equipment list includes the following:

ITEM	NUMBER OF UNITS
27 Ton Eimco Trucks (4 wheel drive)	7
5 Cu. Yd. Eimco Scoops	4
2 Boom Electro/Hydraulic Jarvis Clarke Jumbos	2
1 Boom Electro/Hydraulic Jarvis Clarke Jumbos	1
2 Boom Jarvis Clarke Roofbolter	2
Tamrock Roof Scaler	1
Explosive Trucks	2
Champion Grader	1

Items of general interest regarding the operation are:

- Stope and development production is achieved with the two 2 boom jumbos, the single boom jumbo being held in standby.
- The 'in-stope' productivity of the 2 boom jumbo is a 110 hole pattern with 4.3m (14 ft.) holes drilled in 3 hours giving a 'per boom' drilling rate of 1.3m/min. (4.3 ft./min.).
- Truck loading takes approximately 6 minutes for 4 bucket loads.
- Trucks operate over an average elevation difference of 180m (590 ft.) at speeds of 16 km/hr. down and 6.4 km/hr. up the ramp, giving a production rate of 30 tonnes/hr.

On average, stope blasts measure 12m x 4.3m (39 x 14 ft.) deep by 5m (16 ft.) high, resulting in 750 (4827 tons) tonnes of ore. The daily ore production of 1820 (2005 tons) tonnes is therefore achieved by approximately 2.5 blasts per day. The number of areas available for production is 6. This gives a good measure of the flexibility available in the system, a flexibility which is essential if high productivities are to be achieved.

3. CUT-AND-FILL COSTS AND ACHIEVEMENTS, MANITOBA

The costs and productivities at two mines are compared. These are:

- Thompson Mine: a mature mine, extracting a narrow steeply dipping regular orebody at depths of between 300 to 1200m (1000 to 4000 ft.), using conventional extraction methods with access limited to small raises.

- Trout Lake Mine: a newer fully mechanized cut-and-fill operation with ramp access from surface to all extraction areas, working a wide, but irregular, orebody at present to maximum depths of 240m (785 ft.).

Mines tend to have unique costing systems. It is relatively easy to identify total costs, but usually difficult to quantify costs under different categories. It is, therefore, important to clarify exactly what items are included in an individual cost item before comparison can be made between mines.

The costing study was carried out in July, 1985 and is, therefore, correct to December, 1984.

3.1 Thompson Cut-and-Fill Costs

The costs at Thompson are recorded as $/short ton of broken ore extracted. These have been adjusted to $/tonne. Only direct workplace costs have been identified. These costs take the mining process as far as broken ore delivered to a chute in the haulage. Table 1.2 presents a breakdown of the workplace costs giving labour and material components for each of the major mining elements described in section 2.

3.2 Trout Lake Mine Cut-and-Fill Costs

Costs at the Trout Lake Mine are not kept in the same form as those at Thompson Mine. Some adaptation and approximation was necessary. The costs obtained are presented in Table 1.3. A paper by McIntosh, (1) also gives information on costs and shows good agreement with the adapted costs.

TABLE 1.2 THOMPSON CUT-AND-FILL COSTS

WORK ELEMENT		LABOUR TIME/ELEMENT %	LABOUR $ $/TONNE ($/S.TON)	MATERIALS $ $/TONNE ($/S.TON)	TOTAL $ $/TONNE ($/S. TON)
Ore Breakage;	Drilling	12.3	(1.37) 1.51	(0.55) 0.61	
	Blasting			(0.93) 1.02	(2.85) 3.14
Rock Support;	Drilling	20.1	(2.24) 2.46	(0.30) 0.33	
	Bolting 8'			(0.84) 0.92	
	Bolting 16'			(0.05) 0.06	
	Screen			(0.17) 0.19	(3.60) 3.96
Ore Loading;	Diesel	12.1	(1.35) 1.49	(0.23) 0.25	
	Mtce.			(1.50) 1.65	
	Tires			(0.22) 0.24	(3.30) 3.63
Timbering;	Fill Prep. Raises etc.	11.6	(1.29) 1.42	(0.33) 0.36	(1.62) 1.78
Utilities, Supply Transport; etc.		19.0	(2.12) 2.33	(1.55) 1.71	(3.67) 4.04
Unproductive;	Supervision, Lunch, Travelling Waiting, etc.	24.9	(2.78) 3.06		(2.78) 3.06
Re-Allocated Costs					(0.38) 0.42
Sandfill					(0.85) 0.94
		100.0	(11.15) 12.27	(6.67) 7.34	(19.05) 20.97

These are Direct Workplace Costs. Fringe Benefits and Bonus for Labour are included.

TABLE 1.3 TROUT LAKE IN-STOPE COSTS

WORK ELEMENT	LABOUR COST $/TONNE	($/S.TON)	MATERIALS/ EQUIPMENT COST $/TONNE	($/S.TON)	TOTAL IN-STOPE COST $/TONNE	($/S.TON)
Ore Breakage	0.78	(0.71)	2.19	(1.99)	2.97	(2.70)
Rock Support	0.10	(0.09)	0.23	(0.21)	0.33	(0.30)
Ore Loading (LHD)	0.33	(0.30)	1.87	(1.70)	2.20	(2.00)
Utilities Etc.	0.54	(0.49)	0.30	(0.27)	0.84	(0.76)
Backfill	(Fill is trucked to the mine giving high cost of $1.60/Tonne, (McIntosh, (1) Thompson value used).				0.94	(0.85)
	1.75	(1.59)	4.59	(4.17)	7.28	(6.61)

A direct comparison between Table 1.2 and Table 1.3 is not valid. The Trout Lake mining system achieves low 'in stope' costs and high productivities through intensive mechanization. To achieve this level of mechanization suitable ramp and level access must be provided and considerable capital and replacement finance must be provided for equipment. These are assessed below:

- Development Costs; in 1984 the total development costs were $4.8/tonne ($4.36/s.ton) of ore. Development was however being mined at a higher rate than required for replacement ore by the ratio of:

$$\frac{\text{82 000 tonnes (required for replacement)}}{\text{125 000 tonnes (actual development)}} = 0.66$$

This factor lowers the actual cost to the workplace to $3.17/tonne ($2.88/ton).

Ramps at Trout Lake are used for access and for ore transport. The 4.3m (14 ft.) x 5.5m (18 ft.) ramps are driven at a 13% gradient to enable the use of 2 wheel drive haulage trucks for ore transport.

A valid comparison between the two properties would assume the ramp is used for access only; in which case the ramp could be mined at a gradient of 20% and a size of 3.4m (11 feet) by 4.0m (13 feet).

The Trout Lake adjusted developed cost of $3.17/tonne ($2.88 ton) consists of both incline ramp and access into stopes. A reasonable split on quantities would be one third stope access and two thirds inclined. If we leave the access the same at both mines then the incline portion of the development cost would require an adjustment for size and gradient.

Adjusted Development Cost = $3.17/tonne

1/3 Stope Access = 1/3 x 3.17 = $1.06/tonne

2/3 Incline = 2/3 x 3.17 = $2.11/tonne

Gradient 13% to 20% = 13/20 x 2.11 = $1.37

Size 4.3m x 5.5m to 3.4m x 4.0m = 0.78* x 1.37 = $1.07

TOTAL DEVELOPMENT COST THOMPSON RAMP = 1.06 + 1.07 = $2.13/TONNE

* The cross section adjustment factor is based on time and motion studies at the Thompson Mine and is in agreement with published data (ref. 9).

• Equipment Costs; the high level of mechanization imposes a capital and replacement burden on a workplace. The equipment used at Trout Lake was listed in Section 2.2.3. Equipment relevant to direct workplace costs and in excess of current equipment in Thompson cut-and-fill stopes has an approximate capital value of $1,950,000. If the equipment life is assumed to be 12 500 hrs. the average replacement period would be 5 years and the following additional ownership costs would be considered appropriate:

- Finance cost at 10% = $0.43/Tonne

- Replacement costs = $0.43/Tonne (50% tax rate assuming tax paid)

$0.86/Tonne

The comparable workplace costs now become:

Direct Workplace Costs = 7.28 (in-stope) + 2.13 (development) + 0.86 (equipment) = $10.27/Tonne ($9.34/s.ton)

This is still a relatively unfair comparison as the average orebody thickness at Trout Lake is 12m (39 ft.) compared to less than 5m (16 ft.) at Thompson. The difference in thickness would reduce productivity and increase the amount of access per tonne of ore developed. Conservatively, the maximum estimated cost difference would therefore be:

- 'maximum' equivalent costs 5m (16 ft.) wide ore = $10.27 x 12/5 = $24.65/Tonne ($22.37/Ton)

Trout Lake do have areas of narrow ore and therefore, have some experience of the impact on productivity. In discussions with mine personnel, it was considered that an appropriate cost difference could be derived by doubling the labour complement and directly increasing the development requirement:

- 'minimum equivalent' costs 5m (16 ft.) wide ore = (1.75 x 2) + (4.59) + (.86) + (.94) + (2.13 x 12/5) = $15.00/Tonne Ore ($13.61/Ton

3.3 Thompson Cut-and-Fill Productivity

The cut-and-fill stopes at Thompson tend to operate on single shifts with 3 men crews. The average production per stope is in the region of 76 tonnes (84 tons) per shift giving an 'in stope' productivity of 25 tonnes (28 tons) per manshift. Table 1.4 gives a breakdown of the time spent per shift. It is important to note that only 24.4% of the total time is spent in actually breaking and moving the ore.

TABLE 1.4 - THOMPSON, CUT-AND-FILL TIME DISTRIBUTION BY WORK ELEMENT

WORK ELEMENT	TIME DISTRIBUTION %	TOTAL MAN HOURS
Ore Breakage	12.3	2.95
Rock Support	20.1	4.82
Ore Loading	12.1	2.90
Timbering	11.6	5.64
Utilities Etc.	19.0	1.70
Unproductive	24.9	5.98
	100	24.0

It is customary when carrying out method assessments to take into account unavoidable delay (unproductive time) by assuming 50 min. hours and 7 hour shifts. This constitutes 27% of the available time and therfore the 24.9% of unproductive time in table 3.3 is not unexpected.

3.4 Trout Lake Cut and Fill Productivity

An assessment similar to Table 1.4 is not available for Trout Lake Mine. The mine operates on 3 shifts producing 1820 tonnes (2000 tons) per day. The number of people employed at the mine during 1984 was 78. This results in an overall productivity of 23 tonnes (26 tons) per manshift.

The 'production' crew is considered to be 14 men giving a productivity of 43 tonnes/manshift. Of this crew, four are identified as 'face' and, 6 as 'mucking' crew members. This suggests a direct stoping productivity of 61 tonnes per manshift for 10 men.

Even this crew is not comparable to the 3 man crew within a Thompson cut-and-fill stope as they also do all the development required for access.

An estimation of the 'in stope' productivity can be made from the following information:

- 10 direct mining crew produces; 607 tonnes ore, 162 tonnes waste (development),
- sub-development cost approximately $360/m , giving $13/tonne,
- direct stoping cost (less backfill etc.) is approximately $6/tonne,
- direct mining crew can be distributed between ore and waste on the basis of breaking cost and tonnage broken:

  - ore tonnes = 607
  - equivalent waste tonnes = $\frac{162 \times 13}{6} = 351$

  - giving: 6.3 men on ore production
    3.7 men on waste development

- productivity 'in stope' becomes = 96 Tonnes/Manshift (106 tons).

4. CUT FILL REVIEW - INTERNATIONALLY

A detailed study of cut-and-fill operations around the world is beyond the scope of this review; the objective being rather to identify modern operations which use alternative procedures or equipment that could improve productivity or decrease costs in Manitoba mines. It was evident early in the review that Trout Lake Mine compares favourably with any operation in terms of equipment, procedures and productivity.

Valid comparisons between operations is difficult as it is often not clear what is included in the published production statistics. Also, productivity can vary according to employment practices and labour restrictions.

Given the above constraints, the following critical elements of a cut-and-fill operation are reviewed:

- drilling and blasting
- rock support
- ore removal
- backfilling
- stope preparation.

4.1 Drilling and Blasting

The ore is conventionally broken by drill and blast methods. Alternative rock breaking systems have been considered, such as continuous miners and impact breakers, but none have been brought to a production scale in hard rock.

Within a drill and blast system, the following significant changes could be made:

- increased labour productivity by replacing handheld jackhammers with small jumbos. An equivalent decrease in cost is ,however, unlikely to be realized unless the jumbo can be transported between cut-and-fill faces. This would require ramp access (see Section 4.6).

- decreased powder factor; Powder factors commonly are in the region of 1.0 to 1.2 kg./m ; Trout Lake, Thompson and Black Mountain (ref 1). In contrast, the powder factor at Mt Isa (ref 2) averages from 0.18 to 0.. kg/tonne (approximately 0.6 kg/m) for jumbo and jackhammer operations. This value compares well with the efficiencies obtained in surface blasting and illustrates the potential for improvement.

4.2 Rock Support

The major change which could be made is between point anchored and grouted bolts. In the context however of the present discussion, this would have a minimal effect on productivity or cost. Another possible change which could be considered is the use of a rockbolting rig; again this assumes the presence of ramp access. From the literature the following statistics are relevant:

- Mt Isa (ref 2) quotes 25 rockbolts per manshift unmechanized, whilst Trout Lake Mines obtain 65 bolts per shift with a rockbolter. A cost comparison might result in the following:

  - Cost/installed bolt with bolting rig

Labour, 8 hrs. at $27/hr. (all costs)	=	216
Bolter, 6 hrs. at $50/hr.	=	300
		516
Cost/Bolt, 65 Bolts/Shift	=	$7.94/Bolt

- Cost/installed bolt with jacklegs
  Labour, 8 hrs. at $27/hr. = 216
  Cost/Bolt, 25 Bolt/Shift = $8.64/Bolt

Consequently, there appears to be very little difference in costs, even though labour content has been markedly decreased.

4.3 Ore Removal

The change from slusher to LHD operation has already taken place in most operations. The most significant change would be from captive machinery to ramp access giving much higher utilization. This would also enable the use of larger capacity LHD's. Again, ramp access is the key and is dealt with in Section 4.6.

4.4 Backfill

Backfill is normally placed hydraulically, as has been assumed in the following discussion. The time taken for sufficient consolidation of the backfill to allow mining operations to commence on the surface of the fill is the main control in cost and productivity. The mechanics of filling are sufficiently researched not to offer any likelihood of significant differences in cost or productivity.

There is a considerable move to high density fills of over 75% pulp density. Such fills can make greater use of the available material, are stronger, and are more effective in rock mass stabilization. High density fills have, however, an inherently low percolation rate and usually require the addition of cement to hydrate the entrapped water before consolidation can occur.

This results in a product that is either more expensive than uncemented fills, or the time taken for drainage is much longer. Although there is an operation in Namibia, Tsumeb, which uses a high pulp density backfill in a cut-and-fill operation.

The strength of the backfill in a cut-and-fill operation is not a major consideration as the 'open' excavation is always kept very small. Therefore the cost of strengthening the fill is unlikely to see commensurate decrease in operating costs.

The lack of potential for further productivity or cost optimization in backfilling is apparent from the literature review.

4.5 Stope Preparation

'Stope Preparation' includes such items as:

- catwalk construction to carry services and provide access during backfilling: The 'catwalk is not a standard feature of cut-and-fill operations and is not in fact possible when 'tight' fill is required for ground control purposes. However it does enable very long pours over the full strike length of the stope. This would not be possible without a 'catwalk'. In a fully mechanized system using drill jumbos with ramp access 'tight' fill would be necessary, as working from a muckpile would not be possible.

- raising manways: Ramp access obviates the necessity for manways in the fill

- footwall ore-pass advance: Footwall ore-pass advances would remain

- installation and extension of drainage for backfill: Drainage requirements remain unchanged unless high pulp density fill is used.

The literature review did not reveal any alternative procedures which would significantly alter costs or productivity. It is considered that the requirement for additional fill dams, due to the extended strike length, would cancel the savings realized in reduction of timbering for manway extension.

4.6 Ramp Access

The common denominator between the various changes which could be made, if the conventional cut-and-fill system is to be improved, is the presence of ramp access to every cut-and-fill face. Ramp access is essential for full mechanization. The following information is therefore presented within the context of mechanization:

- raise access versus ramp access, with additional change from jackhammers to jumbos:

  - **Mt Isa Mines (3),**
    Productivity from conventional section = 30t/manshift
    Productivity with ramp access = 40t/manshift

    increase of 33%

  - **Bruce Mine (5),**
    Productivity (total mine) conventional = 7t/manshift
    Productivity with ramp access = 12t/manshift

    increase of 70%

- conversion from jackhammers and slushers to 2 x boom jumbos and LHD's:

  - **NBHC and Zinc Corp (4),**
    Productivity conventional staging = 37t/manshift
    Productivity mechanized = 57t/manshift

    increase of 54%

- Boliden; are probably the most advanced in cut-and-fill techniques and a useful comparison can be made between their Udden Mine (ref 6) and the Trout Lake Mine.

  - **Udden Mine Productivity** = 24 tonne/manshift (full mine complement)
  - **Trout Lake** = 23 tonne/manshift

Further interesting statistics from Boliden (ref 7) in 1982;

- mining cost of blasthole stoping was 25% less than cut-and-fill
- waste rock dilution from blasthole stoping was 20% as opposed to 10% from cut and fill.

At Boliden Mines, virtually all ore is accessed by ramp, whatever the geometry of the orebody.

4.7 Conclusions

The literature review did not identify any new or unusual procedures which are not already in use in Manitoba, or have not been considered or tested. It did demonstrate that mechanization can result in very large increases in productivity. However increased productivity does not necessarily result in a commensurate decrease in costs, as the cost of ramp access and equipment has to be matched by a decrease in labour costs.

For example, an 'in stope' productivity of 27 tonnes per manshift at $27 per total hourly cost, must increase to 77 tonnes/manshift, to cover the cost of providing an individual with equipment to fully mechanize all his activities, assuming an equipment rate of $50 per hour. This simplistic approach is borne out in the optimization analysis in the succeeding section. This analysis, of a fully mechanized narrow orebody system, demonstrates that there is not a significant potential for cost decrease over the present conventional rise access system. Nevertheless, labour costs tend to raise at a faster rate than equipment costs and tonnes per manshift will assume ever greater importance in the future.

The most interesting statistic that came from the literature review is the blasting efficiency at Mt Isa. This requires further investigation to confirm the procedures and results.

5. CUT AND FILL EXPERIMENTATION: THOMPSON MINE

Considerable effort at Thompson has been put into investigating the technology of cut-and-fill to improve the productivity and economics of the method. Various systems have been either tried or analysed, from the conventional to the extremely unconventional, with varying degrees of success. It is pertinent to briefly discuss the results. These must be viewed within the context of a single operation, and the inevitable constraints of their historical procedures and aspirations.

Changes which have had an extensive underground trial are:

- uppers drilling
- mechanized 2 boom jumbo.

Additional systems which have been analyzed in detail are:

- modified uppers drilling
- horizontal long-hole mining
- ramp access within stope.

5.1 Uppers Drilling

The ore was drilled with 3.4m (11 ft.) long holes at an inclination of 60-65 degrees into the back, as illustrated in Fig. 1.19. Heavy duty Joy Wagon drills were used for this method from 1968-76. The advantages of the system were that the cycles of drilling, roof conditioning and blasting did not interfere with one another. Screening and bolting could be done ahead of the drilling, and the uppers drilling could be done well in advance of the blasting and mucking. Achievements were reasonable at 83 (92 tons) tonnes/shift and 21 (23.5 tons) tonnes/manshift.

Drawbacks with the system were:

- the sandfill floor had to be 3.4m (11 ft.) below the lowest point in the back. The resulting 6.4m (21 ft.) back height caused ground problems.
- the unpredictable nature of the orebody dip resulted in breakage beyond the ore contact and caused instability.
- the backfill quality had to be high with a low slimes content otherwise movement of the wagon drills was difficult.

5.2 Mechanized 2 Boom Jumbo

A 2 boom Gardner Denver diesel jumbo was used between 1967 and 1971 in a 30m (100 ft.) wide stope. The stope was mined as a series of 4m (12 ft.) wide faces with 6m (18 ft.) transverse pillars in between. Achievements were 117 (130 tons) tonnes/shift and 26 (28.9 tons) tonnes/manshift. Multiple ends were not available and the jumbo was under-utilized. The main reason for the method being shelved was continuing high maintenance costs for the jumbo.

5.3 Modified Uppers Drilling

Fig. 1.20 illustrates a modification to the uppers drilling method, where:

- a higher face, 5m (15 ft.), could be taken which would decrease the unit costs for bolting and screening.
- longer holes would allow pre-support of the back and inhibit relaxation.

Achievements of 96 (107 tons) tonnes/shift and 25 (28 tons) tonnes/manshift were indicated. The pre-support may have overcome the ground control problems experienced with the uppers method but would not have helped with the variation in dip.

5.4 Horizontal Longhole Mining

Figs. 1.21 and 1.22 illustrate a method conceived in order to:

- reduce the amount of stope excavation in which men would be required to operate

- reduce roof conditioning requirements.

The method was based on longhole horizontal drilling and remote LHD's. Advance raises would be required approximately 36m (118 ft.) apart. The raises would be positioned 10 to 15m (30 to 50 ft.) away from the orebody and have a dual purpose:

- access to the workplaces for men and materials

- after finishing a cut, a concrete bulkhead would be constructed, converting the raise into an orepass.

Mining would start from a raise, advance to the stope where one round would be mined on strike in each direction. The stope excavations would be screened and bolted. A DH 123 drill would be mounted on a remote controlled scissor's lift truck and would drill 50mm (2 inch) diameter holes in a horizontal fan pattern approximatley 15m (50 ft.) long. Once blasted, the ore would be mucked with a remote control scooptram with operator positioned in the conditioned excavations. On completion of mucking the scooptram would be transferred to the next raise.

The analysis indicated potential achievements of 176 (192 tons) tonnes/shift and 36 (40 tons) tonnes/manshift. Although it indicatedpotential, there were objections to the high initial development cost, safety of the crew working over an open chute and the possibility that crews might travel under open ground.

In any event, the method would only have been feasible in very competent ground; based as it was on the assumption that no support was required in the fan drilled areas.

5.5 Ramp In Stope

It was recognized that utilization of equipment in a mechanized system could only be significantly improved if the equipment was not captive. This pre-supposes ramp access and considerable advance waste development. A configuration which minimizes waste development, suggested by Pugh (8), placed the ramp within the fill. This required an approximate orebody geometry of at least 200m (670 ft.) on strike, and a mineable width over 3m (9 ft.) and a steep dip. The method is illustrated in Figs. 1.23 and 1.24.

The method was based on a ramp from the haulage drift into the orebody with a positive gradient of 15 to 20%. The initial cut would be silled out and the second cut breasted down before laying sill timber and filling the initial cut. Subsequent cuts would be blasted onto the fill. Slices would be 4.3m (14 ft.) high and an inclined barricade of 250mm (10 inch) caps and 75mm (3 inch) lagging would be constructed and would form the haulage floor of the ramp.

Analysis of the method indicated potential achievements of 141 (157 tons) tonnes/shift and 66 (73 tons) tonnes/manshift. Although substantial productivity improvements may have been possible, there was no suitable site for a trial; especially as cut-and-fill was rapidly being superseded by the vertical bulk mining method.

5.6 Conclusions

Thompson Mine have concluded that the characteristics of the orebody - very narrow with a low tonnage reserve per metre of strike - do not lend themselves to conversion to a more cost effective cut-and-fill system than is presently in operation.

6. OPTIMIZATION OF CONVENTIONAL CUT-AND-FILL

Optimization of cut-and-fill stoping can be achieved in two different ways. Either adjustments can be made within the particular system of cut-and-fill stoping used at a mine (for example changing the powder factor) or by completely re-vamping the system (e.g. by introducing mechanization).

The potential for carrying out such optimization at the Thompson mine is discussed against the background of an operation which has been going through this type of analysis for many years and finally concluded that significant changes could not be made. This conclusion resulted in the search for different methods and culminated in the development of VBM. Not all of the orebodies at Thompson however are conducive to effective VBM extraction.

The following analysis presents a fresh look at cut-and-fill operations. To assist in this process, results are compared to achievements at Trout Lake Mine.

The conclusions cannot be considered as better than indicative of what might be achieved and the sensitivity analysis which is carried out demonstrates the degree of uncertainty.

Two basic constraints exist in any extraction system:

- **cost**: the cost of extraction must be set against revenue.
- **production capacity**: the production from a mining unit (stoping area) will determine how many units are required, and the area of the mine that has to be developed in order to achieve the required mine production.

Cost depends, in part, on productivity. Production capacity and costs are therefore interdependent.

Costs are made up of:

- **consumables** (supplies, explosives) which have a cost/ton: This cost will depend on the efficiency of a particular process such as mass of explosive per ton or, area of roof per bolt.
- **labor and equipment** - have a cost/hr.: The cost/ton will depend mainly on the tons/hr. achieved per unit.

6.1 Consumables

The consumables at Thompson and the associated cost/tonne are given below:

	Consumables Cost/Tonne
• drilling and blasting	1.63
• roof and wall conditioning	1.50
• timbering; access raise, bulkheads, supplies,etc.	2.07
• sandfill	0.94
• diesel, tires, etc.	2.14
Total Cost	$8.28/Tonne ($7.51/Ton)

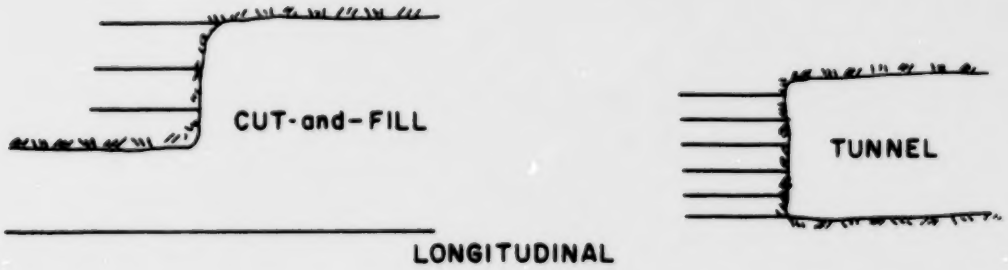
The total consumables cost is 39% of the direct costs. Of this the drilling and blasting is 20%.

6.1.1 Drilling and Blasting

The present blasting pattern results in the following efficiencies:

- **powder factor**: 1.2 kg./m^3 (2.02 lbs./yd^3)

A cut-and-fill blast is completely different from a tunnel round as it has a full free face and no confined toe.



With respect to toe confinement, a cut-and-fill round has easier breaking conditions than a surface blast, but due to the width of the round, more overall confinement.

Typical powder factors for surface blasting with small diameters holes are 0.4 to 0.5 kg./m^3.

Reasons for the higher underground value of 1.2 kg/m^3 (2.02 lbs./yd^3) are:

- **Conservativeness**: historically miners have associated good fragmentation and easy mucking with increased powder factors. This does not necessarily represent good blasting practice.

- **Hole Inaccuracy**: hole direction is very difficult to control with jackhammers and high powder factors may be the response to inaccuracy. But even with jumbo drilling, powder factors in cut-and-fill are often high; examples are Trout Lake Mine, Canada 1.18 kg/m^3 (1.99 lbs/yd^3) and 1.0 kg/m^3 (1.685 lbs./yd^3) at Black Mountain, South Africa.

- **Stemming**: stemming is seldom used in cut-and-fill. It is sometimes argued that the explosive itself acts as a stemming material. The speed of detonation is however very high and the action of the expanding gases is relatively slow. The explosive therefore cannot act as stemming. Effective stemming material must have a high interlocking potential so that it can arch across the borehole and keep re-arching as each arch is broken down. This slows its ejection from the hole. A material which does not re-arch will be ejected as a plug. Angular materials such as coarse drill cuttings are ideal for this task.

A number of additional factors which could be optimized influence the blasting process are:

- **Explosive and Loading**

  Recent cartridged explosives and more efficient loading methods have resulted in significant increases in the amount of power per unit of hole.

- **Pattern**

  The efficiency of breakage increases as the ratio of spacing to burden increases; up to a value of about 8:1. This can be achieved by having a layout ratio of 1.5:1 to 2:1 and tieing up in a chevron pattern to further decrease the effective burden.

- **Drilling**

  Current hole size is 32mm (1 1/4 inch). At Trout Lake Mine the hole size is 51mm (2 inch). The difference in tons per metre of hole for the same powder factor is:

  - 51mm = 2.04 m^3/m of hole (6.5 tons per metre)
  - 32mm = 0.8 m^3/m of hole (2.6 tons per metre)

  Drilling costs tend to increase as a function of the difference in diameter whereas the tonnage broken is a function of the difference in diameter squared.

The above discussion suggests that there is room for improvement in cut-and-fill blasting. Potential savings can be estimated, thus:

- **Potential Savings**

A decrease in powder factor and, an increase in hole size, within the limits of 45mm hole size and a powder factor of 0.7 kg/m , might result in cost reductions of 40% for explosives and 60% for drilling consumables. Labour costs depend on productivity and is dealt with in section 6.2.

6.1.2 Ground Support

The roof has to be supported and this support must be some form of bolt and mesh. Shotcrete as an alternative is not viable on its own as there is considerable stress and strain change. It can be argued that different rockbolts may be more effective and that spacing could be decreased. However, any decrease would be accompanied by a more expensive bolt installation, the savings being in less drilling. The use of 5m (16 ft.) bolts could be dispensed with as they are not nearly strong enough to suspend the weight of rock which they are required to carry at the present pattern. They do not offer greater support capability than a 1.8m (6 ft.) bolt.

- **Potential Savings**

A 40% reduction in bolts required would occur if the pattern is increased from a general standard of 1.44m (5 ft.) per bolt to 2m (6.5 ft.) per bolt (assuming a Q value of 0.2). The bolt cost would likely increase by a similar amount and therefore the materials saving would be entirely in bits etc., (manpower cost discussed later). This is estimated to result in a potential saving of $.12 per ton.

6.1.3 Ore Removal

The present conventional system relies on captive LHDs. Previously, slushers were used but feasibility studies indicated a saving by use of LHDs. This has been justified by experience.

The major problem with captive LHDs is that they can remain in a stope for up to 15 years. This may affect the following:

- **Maintenance Costs**: must be very high. As access is only by raise, travel times are long with repairs often not being effected for at least 2 shifts.

- **Availability and Utilization**: both availability and utili- zation will be low. However, the required operating hours are also low as the LHD is not part of a critical cycle. Drilling and blasting takes place off the muckpile and can proceed without mucking. The average tonnage per shift is approximately 90 tons. This is mucked at approximately 30 tons/hr. with a 2 cubic yard loader; giving a utilization of 40%. Availabilities for non-captive loaders in severe conditions are seldom better than 65 to 70%. It is not difficult to envisage a situation where a captive loader, after many years of operation, cannot achieve the required utilization. Potential increases in productivity would make availabilities even more critical.

It is apparent that savings can only be achieved if maintenance, access, and utilization can be improved. The operational cost for the captive 2 cu. yd. loaders is approximately $43.5/hr., whilst for non-captive loaders it is $41.2/hr..

- **Potential Cost Saving:** $0.10/ton for difference in operating cost. There is unlikely to be any difference in other consumables.

6.1.4 Timbering, Utilities and Backfill Preparation

Backfilling requires a certain quantity of consumables whatever the system used. Drainage and access are provided in the fill in a conventional system. The cost of consumables for drainage is unlikely to change although the type of consumable, may make a difference to labour productivity.

If an external access was provided a reduction in consumables may be possible as some access raises may not be required. The materials cost which would be affected, is only $0.33/ton. Savings in excess of $0.15/ton are therefore difficult to envisage.

6.1.5 Discussion

Total savings in consumables which may be possibly are:

$0.33 Drilling
$0.37 Explosives
$0.12 Ground Support
$0.10 LHD Hourly Costs
$0.15 No Access in Fill

$1.07/Tonne = Possible Total Saving

6.2 Labour and Equipment Productivity

Productivity does not always follow design logic. This is due to the expectations or aspirations, modified by the general experience, of a workforce. Changes can be made which should alter productivity but which in practice do not.

It is therefore imprudent to identify changes in procedures or equipment and quantify the increase in productivity. If it is a new mine which is coming off the drawing board then productivity can be quantified as operators and supervisors do not have pre- conceived views on attainment. This is not the case in a mature operation.

Present "in stope" productivity at Thompson is 25 tonnes/manshift. The time elements are approximately as follows:

TABLE 1.5 - TIME ELEMENTS OF "IN-STOPE" PRODUCTIVITY

WORK ELEMENT	PERCENT OF SHIFT	MANHOURS/SHIFT
Drilling and Blasting	12.3	2.95
Excavation Support	20.1	4.82
Ore Extraction (LHD)	12.1	2.90
Fill Preparation/Timbering	11.6	2.78
Utilities/Nipping	19.0	4.56
Unproductive (travelling, supervision, lunch, etc.)	24.9	5.98
	100.0	24.0 (3 men/shift)

The following productivity assessments are divided into two parts. The first, considers the present cut-and-fill system with the level of mechanization confined to captive loaders and access provided by raises only. The second, considers ramp system provided to give efficient access to workplaces enabling higher levels of mechanization.

6.2.1 Productivity Potential with Raise Access

Without changing the procedures or equipment there is very little opportunity of changing productivity. In the previous section only drilling and blasting, and support showed potential for productivity increases.

The average benching unit for the time study, Table 1.5, contained approximately 100 tonnes. The mine achieved a productivity of approximately 25 tonnes (28 tons) per manshift. A three man stoping crew is therefore able to mine 75% of the basic mining unit in a shift.

- **Drilling and Blasting**: on average, 3 man hours are spent on drilling or blasting during a shift (see Table above). As stated above, some 75% of a basic mining unit is mined during a shift. Therefore, to completely drill and blast a basic unit will take 4 hrs. In the previous section it was suggested that reductions in powder factor and an increase in hole size could reduce the number of holes. The maximum hole size for a jackhammer has been assumed to be 45mm (1 3/4 inch). A 45mm hole will take much longer to drill than a 32mm hole and little time would be saved on hole size alone. A change in powder factor from 1.2 kg/m^3 to 0.7 kg/m^3 could however result in a saving in man hours for drilling and loading fewer holes. If a linear relationship is assumed between the reduction in powder factor and the drilling and blasting time of 2.95 hrs. (table 6.1) then a saving of 1.23 hrs.

or 5% of total time could be achieved. This small increase in productivity can only be achieved if, within the mining unit, it is possible to use a far more efficient pattern which is unlikely.

- **Excavation Support**: it has been suggested that the rock bolt spacing could be increased if fully bonded bolts were used. In most cases fully bonded bolts take longer to install than point anchor mechanical bolts as indicated in table 6.2 below. It is concluded that very little productivity increases are available by using different rock support techniques.

TABLE 1.6 - COMPARISON OF EXCAVATION SUPPORT INSTALLATION TIMES

8' Mechanical Rockbolt	100%
8' Swellex Bolt	107%
8' Resin Bolt	135%
8' Rebar Cement Grouted	182%
10' Cable Bolt Cement Grouted	407%

6.2.2 Productivity Potential with Ramp Access

Ramp access provides three very strong advantages. These are:

- **to allow full mechanization**: mechanization usually means rubber-tired diesel units. These can only be used to their full capability if they can be maintained efficiently.

- **to provide materials access**: movement of materials into the mining area can be much more efficient. A level of palletization is possible in which all the materials for a mining unit can be assembled on a single pallet.

- **to maximize equipment capacity**: the production capacity of mechanized equipment can be utilized if that equipment can be easily moved from one mining area to another.

A ramp is therefore essential for any real increase in productivity to be possible.

A ramp can be mined in the waste or can be mined within the orebody and maintained in the backfill as shown in Figure 1.23. If the ramp is mined in waste there is an extra cost per tonne of ore extracted. Within the ore the cost is equally high but some ore is forthcoming during the ramp establishment and crosscut access developement is not required. The cost of supporting and maintaining the ramp within the fill would be extremely high. Also, the ramp within the fill is likely to contain external support in the form of timber and the conditions may not be ideal for the mechanized equipment. On balance it is considered

that the final costs would not be significantly different between a ramp in waste and a ramp in ore. Further discussion will be confined to a ramp in waste, the conventionally adopted form.

A number of aspects require examination:

- potential increase in productivity for labor and existing equipment
- hourly cost and productivity of additional mechanization; conventionally this equipment would fall within the following activities:
    - **benching**; drilling and blasting
    - **excavation surface conditioning**; scaling, drilling of rock bolt holes, installation of rock bolts and finally, screen
    - **ore removal**; larger LHD's may be required
    - **services**; service vehicle, material transfer, scissors lift for utilities (if required) and screen installation.
- mechanization assumes increased productivity; either this increased productivity is absorbed by reduced crew sizes, or, additional work areas are required so that either larger or more ends are available per shift.
- the presence of an external access system means that no access would be required in the fill and only drain towers would be needed.
- the present practice of carrying the benching activity well ahead of the ore mucking position would not be possible if mechanized drilling by jumbo was used; cycles would become entirely interdependent.

The first step is to identify the different items of equipment, capacities and the potential time and labour requirements for the individual activities.

6.2.3 Mechanized Equipment

The different activities have been identified above. The equipment which could logically address each activity is given below:

- **drilling**: using a self-propelled, diesel/hydraulic two boom jumbo. Typical long term achievements for a two-boom jumbo is three standard development heading per day on a two-shift basis. LA typical powder factor in a development heading would be 2 kg/m^3. Less than half of the number of holes would be required for the same volume of broken material in a cut-and-fill face. The time taken to finish a round however is not directly proportional to the number of holes.

  In a stoping environment a typical advance will contain 100 tonnes of ore. Daily production (i.e. three faces) will therefore be 300 tonnes. If it is assumed that by reducing the powder factor and

the number of drill holes required by half an equivalent increase in production is realized it may be possible to increase the daily production from 300 tonnes to 450 tonnes. A drilling rate of 450 tonnes/day (2 shifts) is achievable with a jumbo. This would represent 4.5 faces of 100 tonnes each. The governing factor would be faces available not the drilling capacity.

- **charging**: the distance from the sandfill floor to the back might be as high as 5 to 7m (7m being the height used at Trout Lake MIne). A scissors lift, or equivalent, would be essential. This would be equipped with cartridge or powder loading equipment.

- **scaling**: scaling is usually done by hand. It is possible that simply impacting on the back with an oversize chisel bit with a jumbo would perform the scaling task more efficiently than at present. The usual problem with this procedure is that damage is done to the boom of the jumbo, either due to levering or rocks falling onto the boom. An alternative is to scale form the back of the scissors lift or, as in the case of Trout Lake MIne, to have boom mounted hydraulic scaler on its own carrier. In the present context is seems unlikely that a separate scaler would be efficiently utilized. Either the drilling jumbo or service vehicle would therefore have to be used.

- **rock bolting**: the rock bolting requirements per tonne of ore produced is approximately 1 bolt per 10 to 15 tonnes of ore. The bolt holes would be drilled by the jumbo requiring some 30 to 45 holes per day, involving an extra 90 minutes of work per day; assuming a drilling rate of 1m/min. for a 2 boom jumbo. Because of movement constraints, bolts needed before drilling from the scissors lift. This problem is overcome in most instances, such as Trout, by having a separate mechanized unit. In narrow orebodies, with smaller workplaces it is difficult to envisage a rockbolting unit being used effectively.

  If the jumbo has to double as a rockbolter then it must be able to drill perpendicular to the excavation surfaces. A telescopic drill carriage will be required and changes in drill steel length will be necessary. It may also limit the round length if boom/carriage length is critical, unless extension steel is used.

- **screening**: handling and erection of screen would only be possible from the service vehicle equipped with scissors lift. Screen would therefore be installed after mucking out and would be within one round distance of the face. During drilling operations the jumbo operator would be positioned under screen, as well as having a reinforced cab roof.

- **ore removal**: productivity, as measured in tonnes/hr., can only be increased by using larger units a 5 cu. yd. size would give optimum efficiency within the size constraints. Theoretical capacity is:

  - 100m average one way haul at 5 km./hr. average speed — 2.4 min.
  - 1 min. loading and tipping time — 2.0 min.
  - 1/2 min. maneuvering — 0.5 min.

  5.0 mins./bucket

  - bucket capacity = 0.8 (fill factor) x 0.65 (swell factor) x 3.2 (density) x 3.8 (bucket capacity) = 6 tonnes giving 72 tonnes per hour.

The actual achievement for a captive 2 cu. yd. loader is:

- Average tonnage per shift = 3 x 27.6 = 83 tonnes
- Average operating hours = 2.9

- Achievement = 29 Tonnes/Hr.

which agrees with the rate calculated for the larger buckets (i.e. 2/5 x 72 tonnes/hr. = 29 tonnes/hr.

The next step in assessing productivity capability and optimization is to fit the above list of equipment into the available workplaces and to assess their interaction, or interference. This will provide an overall tonnage per day, hours worked for each item of equipment, finally, the number of man hours required to operate the units and fulfill other activities.

6.2.4 Stope Productivity

The presence of a ramp access does not, in itself, provide more workplaces unless, within a block, two or more levels are operated between the main haulage level.

An average blast with a jumbo might involve 160 tonnes (177 tons); accepting the average width of 4.6m (15 ft.) and pulling 4m (13.1 ft.) over a height of 2.75m (9 ft.). The height will depend on the width of the blast using a criteria of the height being limited to the width. This is a function of confinement; if the holes are over-confined, fragmentation will suffer and damage to the excavation wall will increase. Since the minimum mining width is in the region of 2.75m (9 ft.) the blast height has been limited to this value.

It has been suggested that in a narrow orebody the maximum tonnage broken by a 2 boom jumbo is likely to be 450 tonnes per day implying an average of 2.8 faces per day, reaching 4.7 faces per day when all areas are being mined at the minimum width. At least two operating levels are required since at times only a single end would be available on a level. A stope divided into two operating levels results in two remnant pillars and increases the underhand cut-and-fill element. This must be taken into account as a reduction in productivity and increase in cost.

The approximate unit times for various activities have been assessed as follows:

• **2 Boom Drilling Jumbo**: giving approx. 3 faces/day	TIME/MIN.
Travel to face and set up	30
Scale	20
Drill rock bolt holes; 20 bolt holes 40m of drilling at 1 min./m.	45
Install rock bolts; 20 bolts	30
Mark up face	15
Drill off face; 12-15 holes, 45-51mm dia.,4m (13.1 ft.) long (Trout Lake achieved 36 holes/hr.	45
	185
Repeat for face 2, 3	185x2

Move between levels and finish 1/2 of next face	137.5
	9.25 Hrs.

- **LHD**: 450 tonnes at 72 tones/rh. (5 yd. machine)

Muck 3 faces plus travel between levels	7 Hrs.

- **Service Vehicle**:

Charging and connecting a face	60x3
Travel	30
Screening	60x3
Travel	30
TOTAL	23.25 Hrs.

This total productive time, or time easily identified with mining advance, is equivalent to 44.5% of the normal mining cycle in conventional cut-and-fill stopes, Table 1.4 One way of estimating total man hours could be:

- 23.25 man hours = 44.5%

  $23.25 \times \frac{100}{44.5}$ man hours = 100% Activity

which is 4 man crew on two shifts (with approx. 8 hrs. slack).

- Productivity = $\frac{450 \text{ tonnes}}{64 \text{ man hours}}$ = 56 tonnes/manshift (62 tonnes/manshift compared with 27.6 at present)

None of the individual units of equipment are overcommitted. It should be noted that potential increase in efficiency for other operations within the 55.5% of the 'non-productive' time have been disregarded. This may, or may not, be conservative. In the view of the authors it is not considered conservative for the following reasons:

- A 'shared' shift using 2 crews results in lost time due to unavoidable friction between the crews.

- It is difficult to cover all the likely problems and delays which are bound to occur within the severe and constricted mining environment of a long narrow cut-and-fill mining geometry. Much of the work may be within minimum mining widths.

- Availability of faces is very difficult to quantify. Although 3 available faces have been assumed this is likely to vary from 3 and 5 and average at 4; adequate for the average mining width of 5m, but his is 41% too low if all faces are within the minimum mining width.

It should be noted that any widening of the orebody beyond 5m would result in an almost direct increase in productivity as the only item of equipment which would be significantly affected would be the LHD.

The final step in this approximate assessment of potential is to assess the cost impact.

6.2.5 Direct Mining Costs

In Section 6.1 it was demonstrated that very little cost benefit could accrue from reduced consumables. It is proposed that consumable costs remain unaffected and the difference in costs relate exclusively to the following:

- Cost of establishing ramp access between levels; charged as a cost per tonne based on the availability and cost of money.
- Cost of cross-cut access to each cut; charged directly to tonnes extracted.
- Cost per hour of labour under the revised structure and productivity.
- Cost of the equipment by the hour.

These allow direct comparison between workplace costs at present and potential workplace costs for the increase productivity. It does not give the total costs as it does not account for the normal pre-production development.

Ramp Access

The length of ramp Figure 1.25 is calculated from the following: - four hundred foot interval between levels; 20% gradient; 15m radius on curves with 10% gradient in the curves to bring the ramp to the next access position. The ramp must rise within a relatively short strike length if it is to confine access within the central third of the stope and provide the number of mining faces used in this study. A schematic of the ramp is shown in Figure 1.25 with the following values:

- Total Length = 555m straight + 188m curves = 743m
- Total Cost, at $250/ft. = $609,445
- Cost per operating tonne with a money value of 10%, operating tonnage of 108,000 tonnes/hr. and a total block tonnage of 480,000 tonnes giving capital cost per tonne of $1.27/tonne and $0.56/tonne interest.
    - Per Tonne Operating = $1.83/tonne.

Horizontal Access

Access on each level at 20% gradient over a lift height of 3m, for each access and roof slash, will be 15m long. Cost of slashing taken as half full face cost. Total access cost for 3 slices = $22,555. Tonnes per three slices = 35,000 tonnes.

- Per Tonne Operating = $0.64/tonne ore; for access.

It is considered that the $250/ft. for development though is very conservative for both the access and the ramp. A higher level of support and utilities etc. then would be normally required have been included, this cost is accepted for this study.

Equipment

- **LHD 5 Cu.Yd. Loaders**: present cost at $57/hr. without operating labour; gives a cost per day = $399/day
- **Drill Jumbo**: assessed at the same cost as LHD but without drilling consumable included; gives a cost per day = $572/day
- **Service Vehicle**: assessed at $15/hr.; 6 hrs. productive time and further 6 hrs. of nipping etc. = $180/day
- **Ownership Costs**: are approximately 30% of the operating cost giving a total ownership cost of approximately = $377/day

TOTAL COST = $3.4/TONNE

Labour

At a base average cost of $14.73/hr. plus 40% bonus and 43% fringes giving an overall rate of $27/hr.; giving a total daily cost of $1728/day.

Total Cost = $3.84/Tonne Ore

Consumables, Materials and Sandfill

These costs are taken from the cut-and-fill cost report section.

Total Cost = $6.14/Tonne of Ore
(excluding diesel, tires, etc.)

6.3 Discussion

The total mining cost is $15.85/tonne (equivalent to $14.38 per ton). At face value this is less than the present cut-and fill cost. It is based, however, on a number of productivity assumptions. The operating costs sensitivity can be assessed as follows:

- **Major Assumption is Availability of Faces:** if this should drop by one third over the life of the stope then the cost per tonne could vary by:

• Ramp Access:	Capital Cost	=	$1.27/Tonne (Unchanged)
	Interest Cost	=	$0.84/Tonne (+50%)
• Horizontal Access		=	$0.64/Tonne (Unchanged)
• Equipment		=	$5.10/Tonne (+50%)
• Labour		=	$5.76/Tonne (+50%)
• Consumables		=	$6.14/Tonne (Unchanged)
			$19.75/Tonne

which brings the cost in line with present cut-and-fill costs.

At Trout Lake Mine the direct comparable mine costs are approximately, $15.0/tonne for an equivalent 5m wide ore zone (see Section 3.2) for the financial year 1984. This sum is fairly close to the value of $15.85/tonne synthesized for Thompson.

A further consideration is the additional tonnage which may have to be extracted by underhand cut-and-fill methods within the intermediate pillar. This tonnage could, however, be extracted by VBM methods at a low efficiency and possibly high dilution. But even as an undercut-and-fill operation much higher productivities would be achieved. An extra 15% of the block tonnage could be tied up in ground that would be too unstable for conventional cut-and-fill. The cost impact on the rest of the block would be in relation to the difference in cut-and-fill and underhand cut-and-fill. The ratio of costs at present is $19.47:$31.81. The total extra cost for the underhand portions:

$$.15 \times 480{,}000 \text{ tonnes} \times [(15.85 \times 31.81) - 15.85] = \$723{,}000$$

$$\text{Extra Cost Per Ton of Total Ore} = \underline{\$1.50/\text{Tonne}}$$

- **Conclusions**

Initial calculated cost for fully mechanized cut-and-fill; = $15.85/Tonne

A one third reduction in assumed productivity = $19.75/Tonne.

Accounting for additional under cut-and-fill extraction = $21.25/Tonne.

The degree of confidence on these calculations is approximately $16.00/Tonne to $21.00/Tonne to be compared with the present cut-and-fill costs of $20.97/Tonne.

This should be viewed in comparison with Trout Lake mining cost adapted to 5m orebody width of $15.00/Tonne in 1984 money.

The number of available faces dictates the viability of applying full mechanization and ramp access to the Thompson orebody.

7. CONCLUSIONS

Mining methods, procedures and geometries of two very different mines, both operating in Manitoba, have been described and compared in terms of costs and productivities Table 1.7. The cost study was carried out in July, 1985 and is, therefore, correct to December, 1984.

TABLE 1.7 - CUT-AND-FILL COMPARISON

ITEM	THOMPSON	TROUT LAKE
1. Average Width	5m	12m
2. Average Productivity	7 Tonnes/Manshift	23 Tonnes/Manshift
3. In Stope Productivity	25 Tonnes/Manshift	96 Tonnes/Manshift
4. In Stope Cost	$20.97/Tonne	$7.28 Tonne
5. In Stope, + Adjustment for Access, Equipment		$10.27/Tonne
6. In Stope, + Adjustment for Orebody Width		$15.00/Tonne

A brief review of cut-and-fill internationally resulted in the conclusion that:

- Trout Lake MIne was the equal in procedures and productivity of any other operations.
- None of the more modern and highly mechanized operations have captive equipment - they all operate with ramp access, no matter how narrow the orebody.
- There is considerable potential to reduce the powder factor, possibly by 40%.

Nevertheless, the review did not suggest any new systems or procedures which have not already been considered or tried in Manitoba. Likewise, it did not indicate that productivities and achievements are significantly different, given the constraints of methods and orebody geometry.

- A fully mechanized system with ramp access was analyzed within the Thompson Mine constraints and costs estimated:

  - Productivity per Stope = 450 Tonnes/Day
  - Labour Productivity (In Stope) = 56 Tonnes/Manshift
  - Cost per Tonne of Ore = $15.85/Tonne
  - Cost per Tonne of Ore with Availability of Faces Reduced by One-Third = $19.75/Tonne
  - Cost per Tonne of Ore Increased for possible Additional Underhand Cut-and-Fill = $21.25/Tonne

In conclusion, the two Manitoba cut-and-fill operations reviewed in this report demonstrate a satisfactory level of cost and productivity within the constraints of the methods. It is evident that much higher levels of productivity could be achieved by providing ramp access for the method at Thompson and by increasing mechanization, but it is uncertain whether a decrease in cost could be achieved.

What the study clearly demonstrates is that cut-and-fill remains a viable method, comparable in achievement with VBM in certain geometrical circumstances

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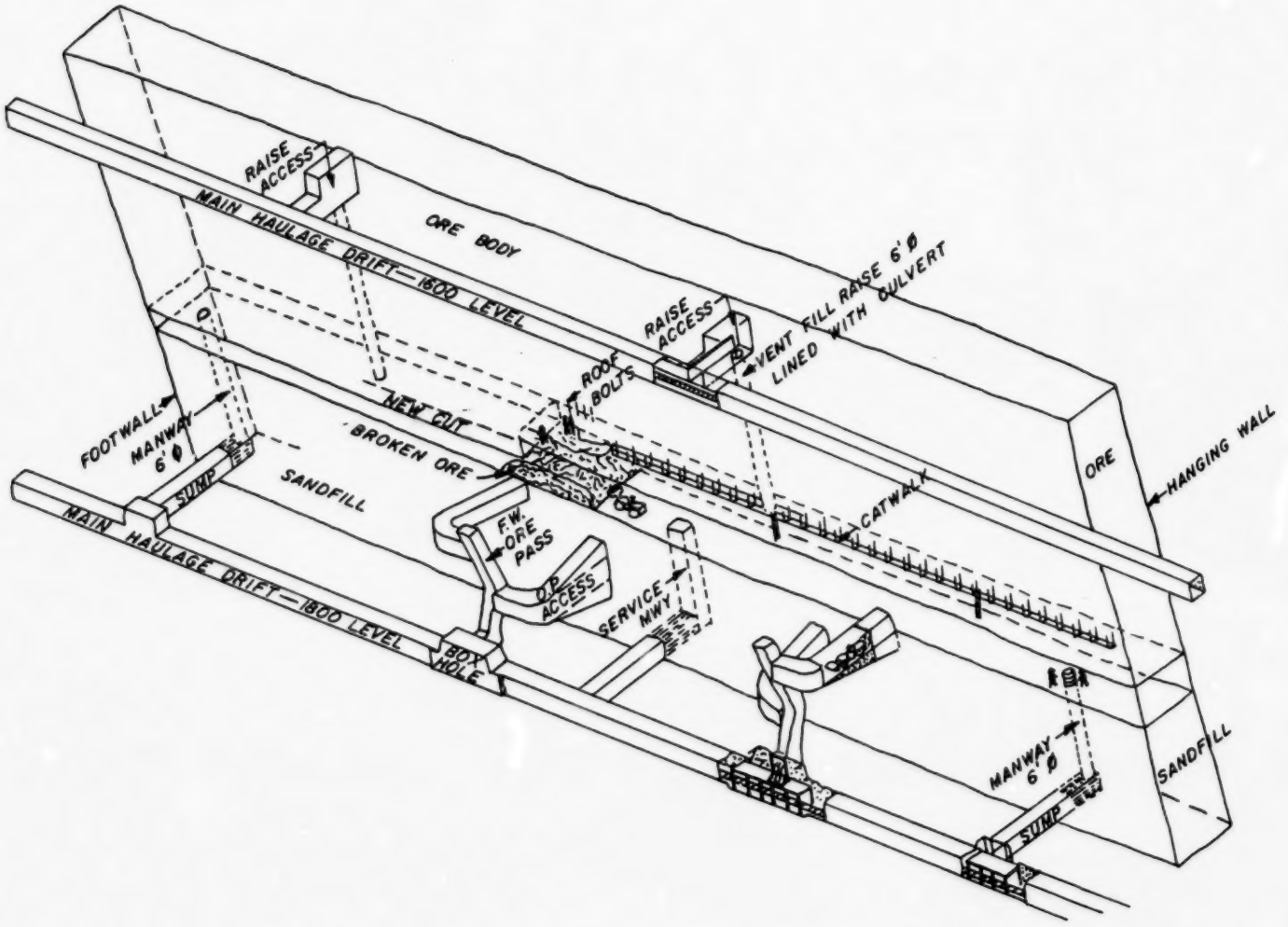
9. Anon



ISOMETRIC VIEW OF A TYPICAL MECHANIZED CUT & FILL STOPE

FIGURE 1.1

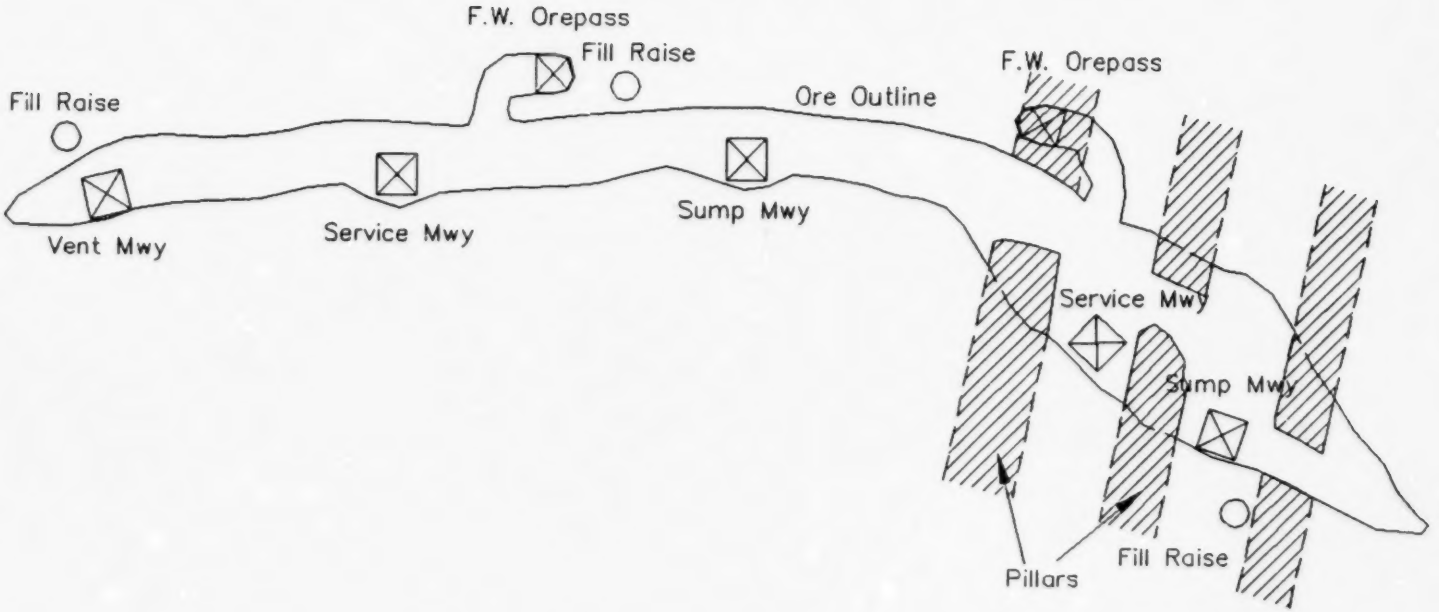
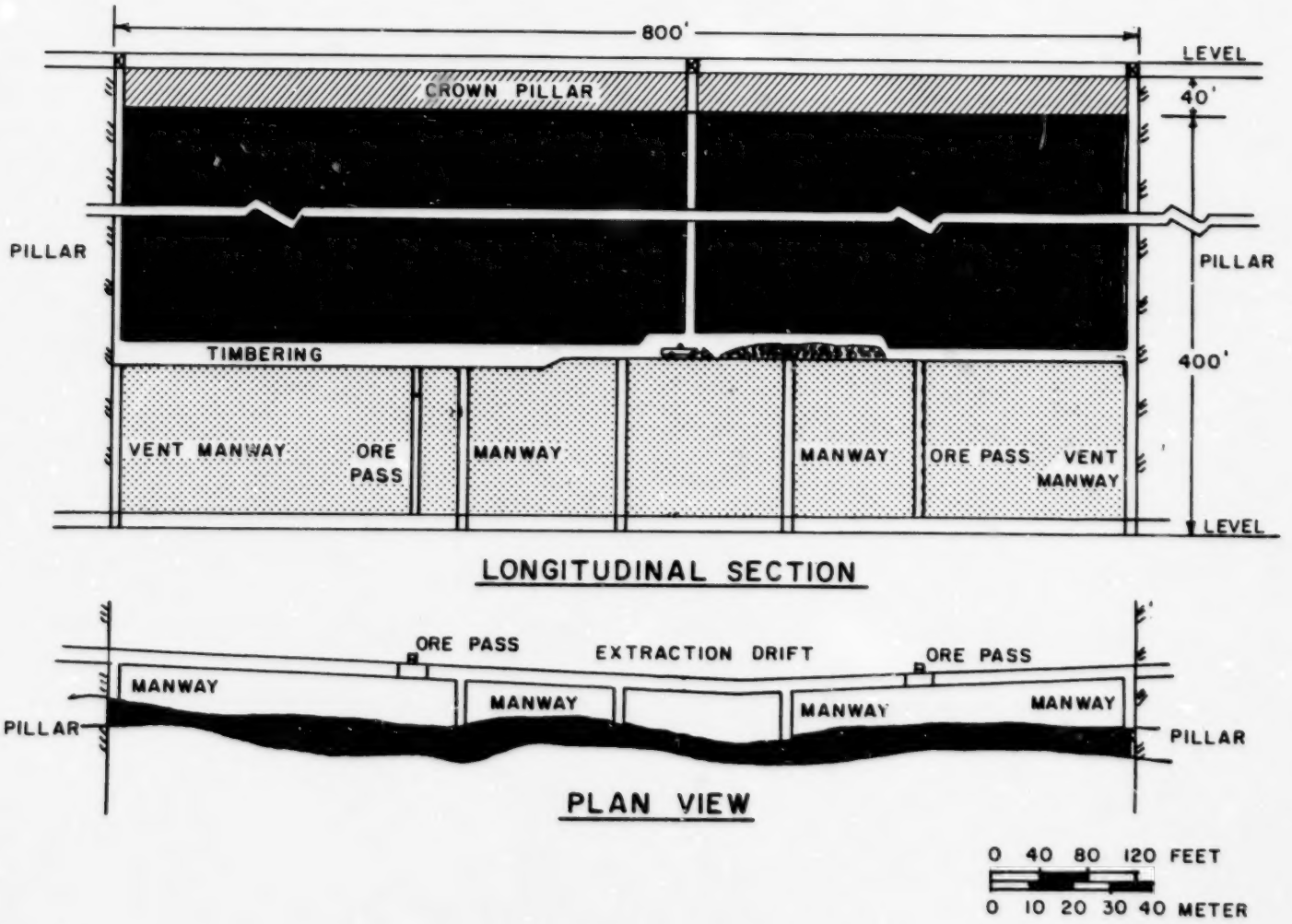


Figure 1.2

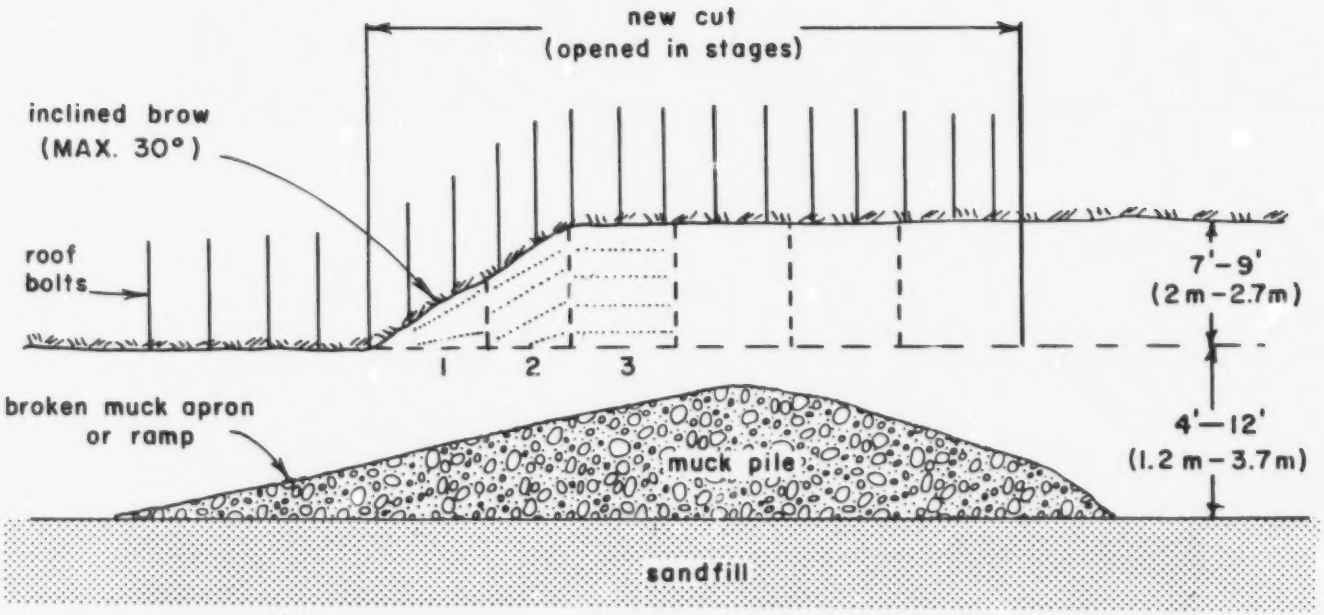
Plan Of a Typical Mechanized Cut and Fill Stope



TYPICAL MECHANIZED CUT AND FILL STOPE

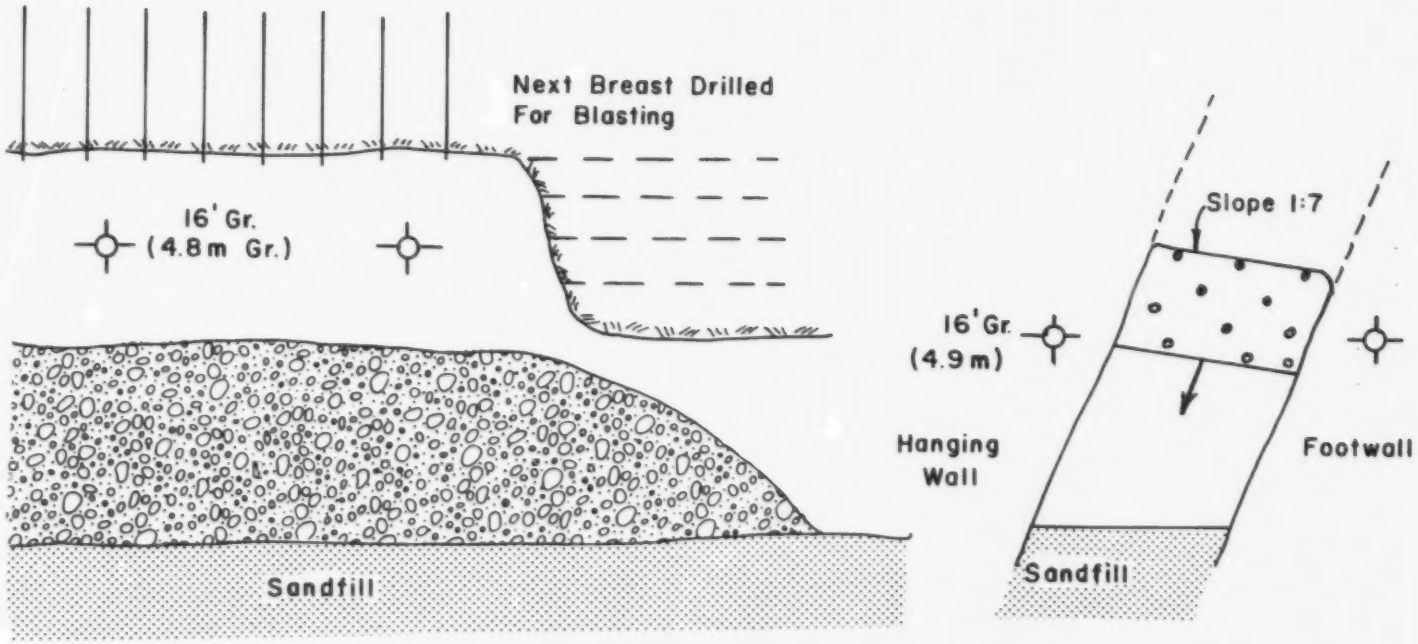
FIGURE 1.3

OPENING NEW CUT WITH INCLINED BROW
(general view — not to scale)



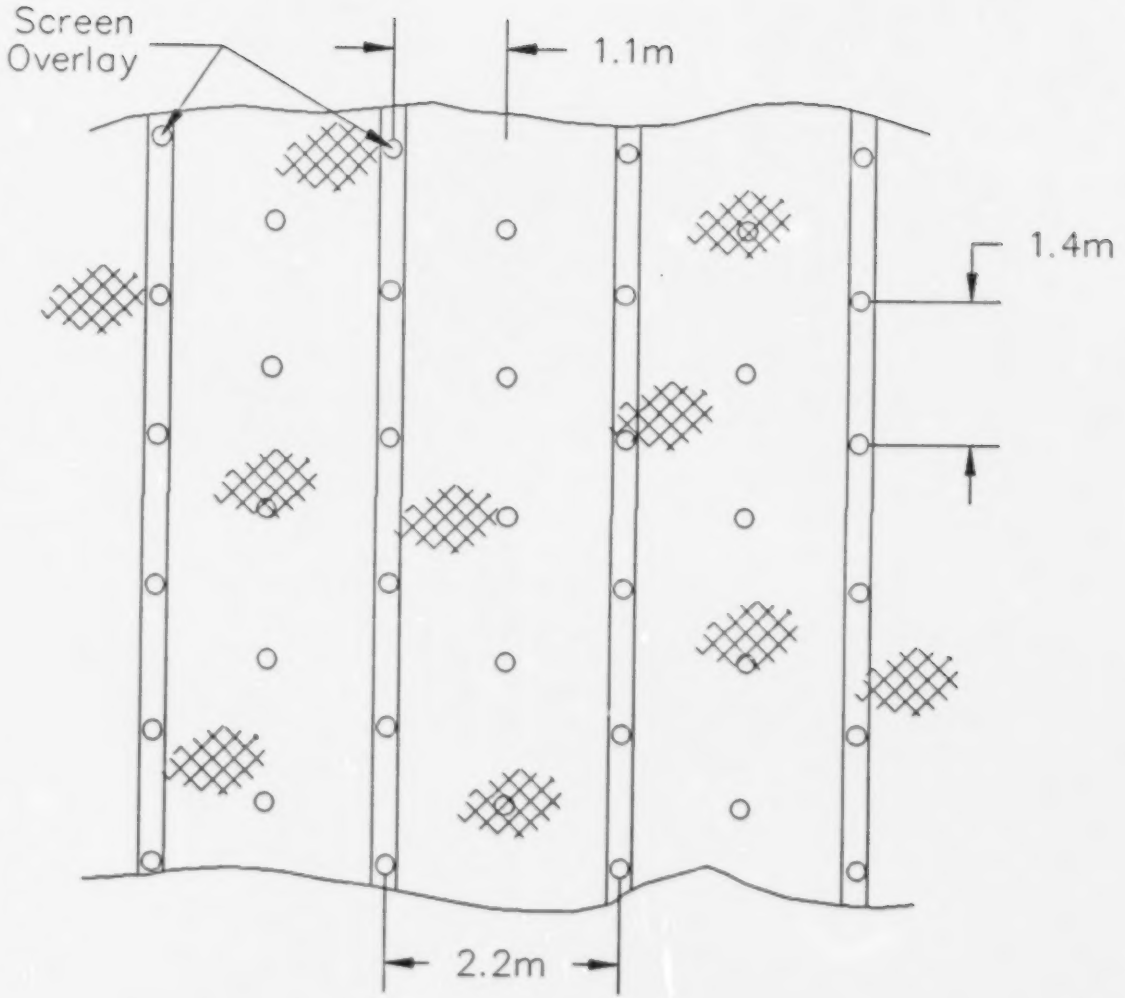
New Cut General Arrangement

FIGURE 1.4



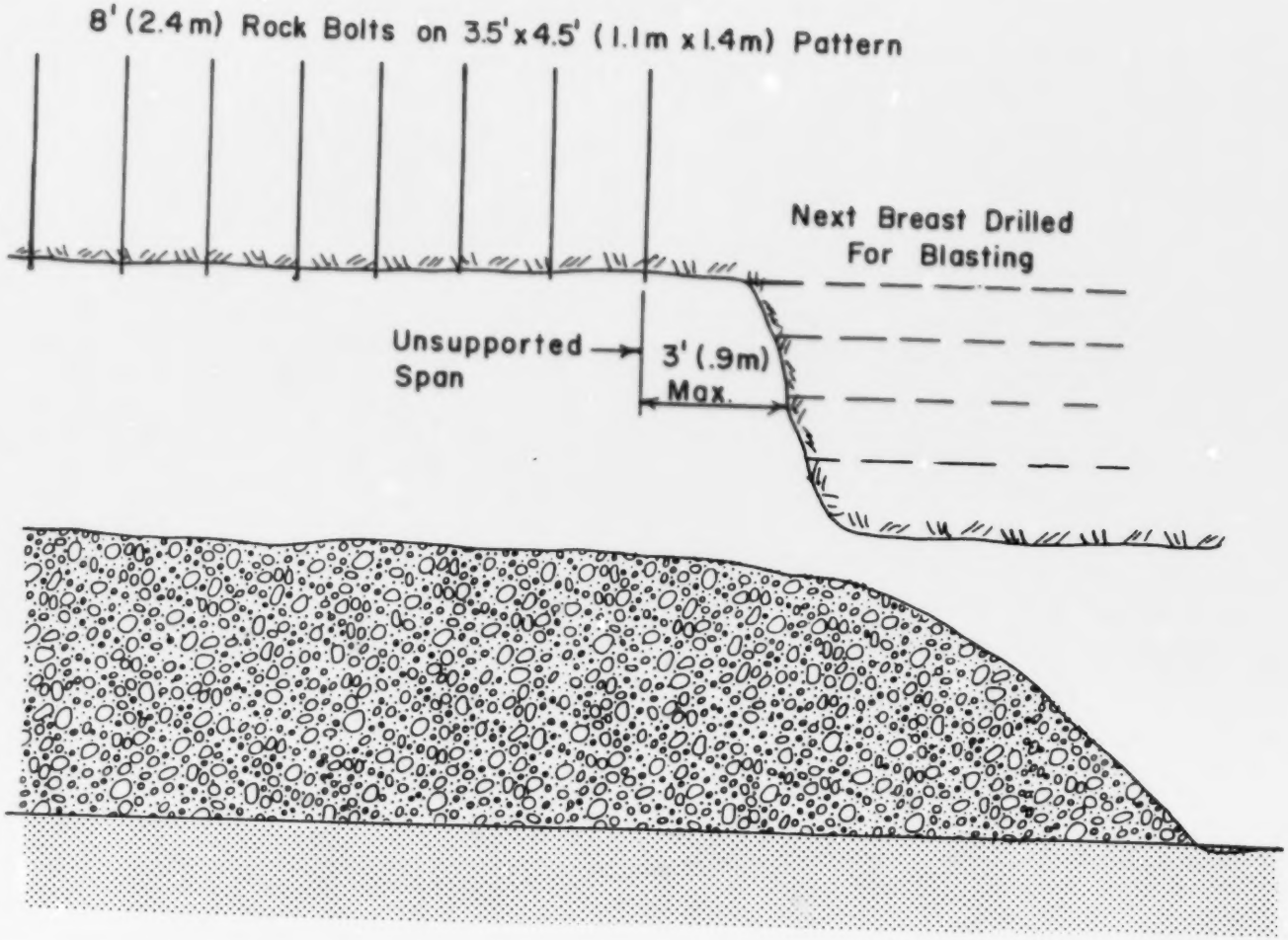
Typical Regular Mining Advance

FIGURE 1.5



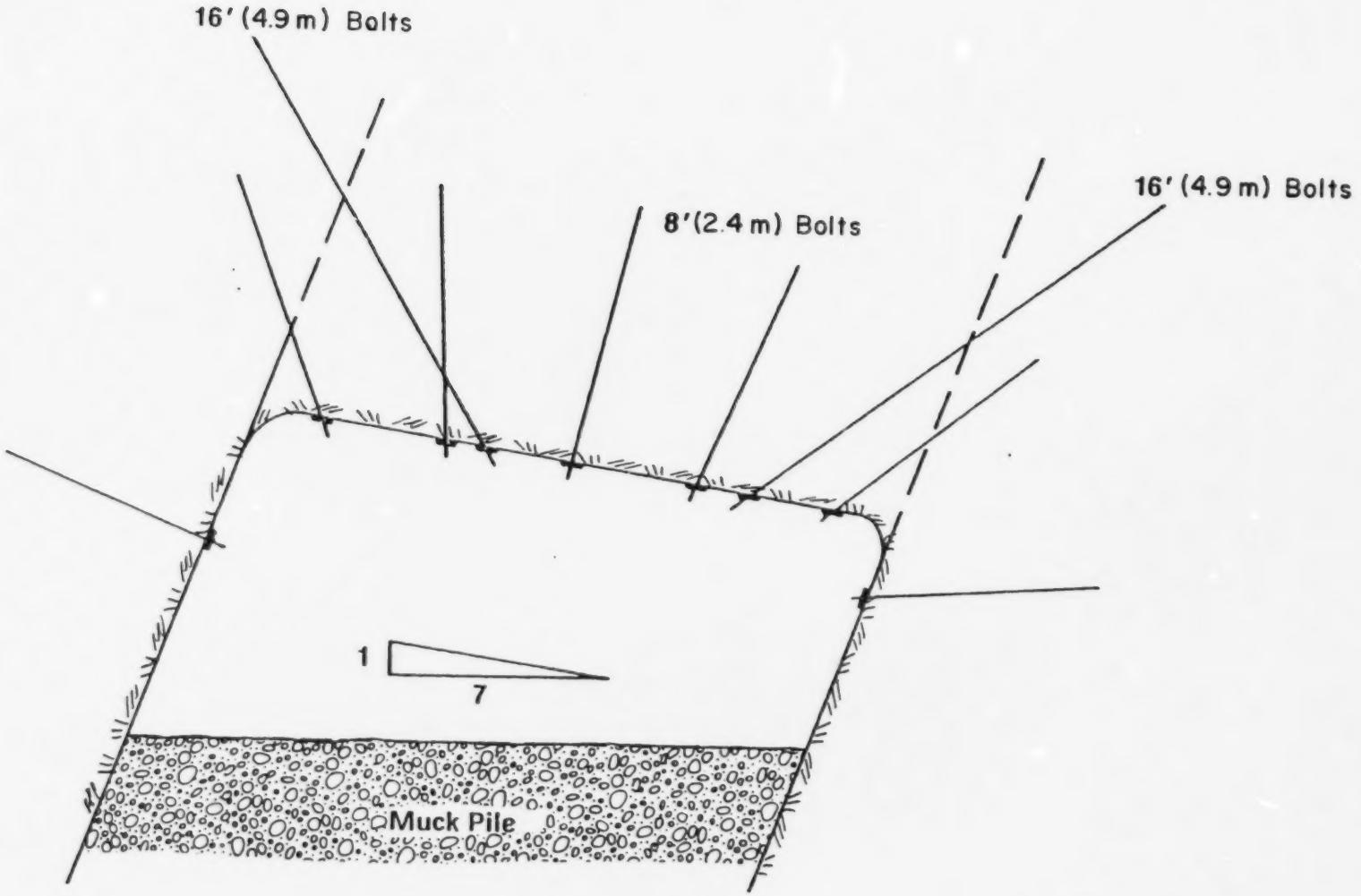
Ground Conditioning - Plan Of a Typical Stope

FIGURE 1.6



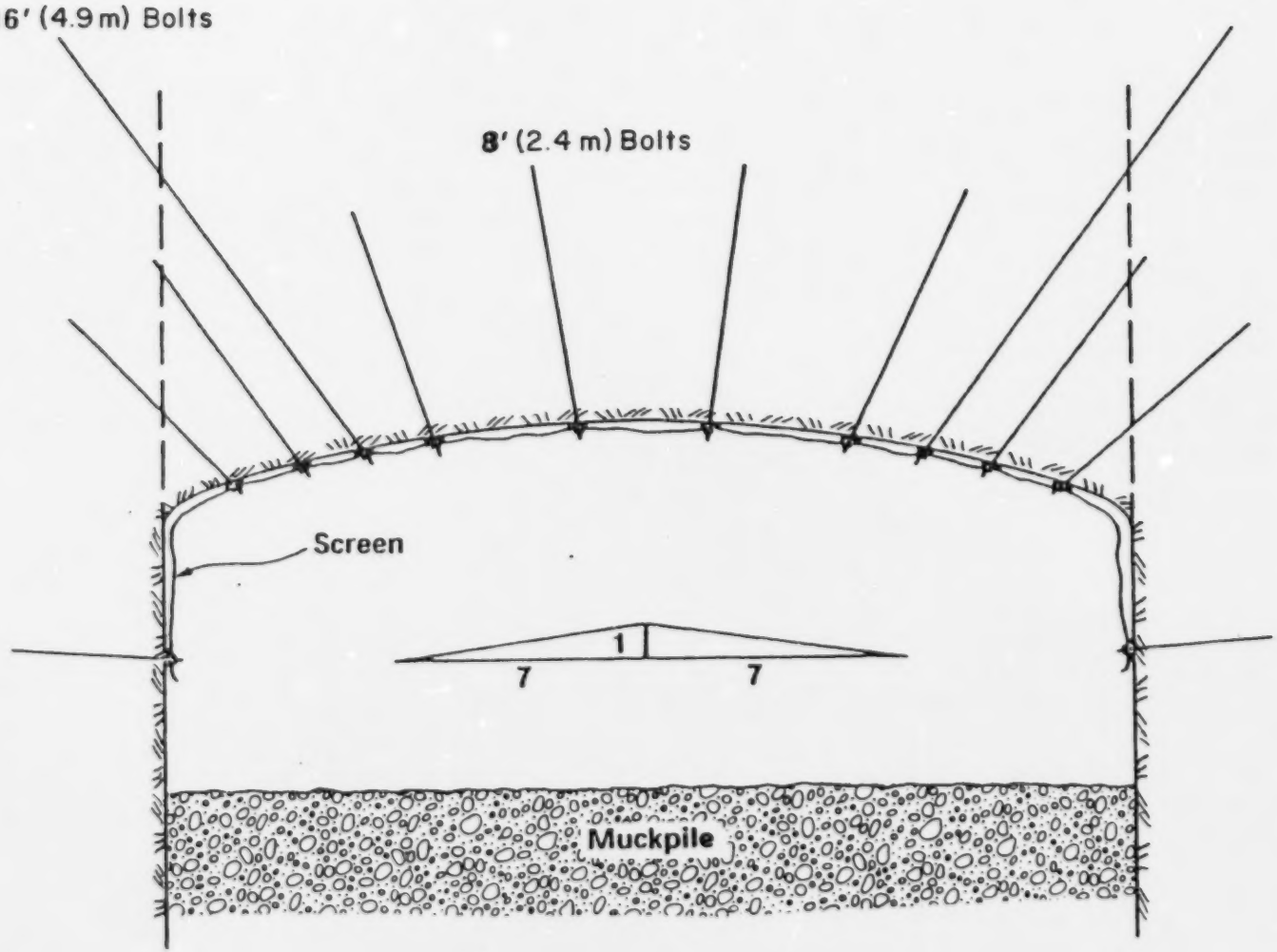
Ground Conditioning – Longitudinal Of A Typical Stope

FIGURE 1.7



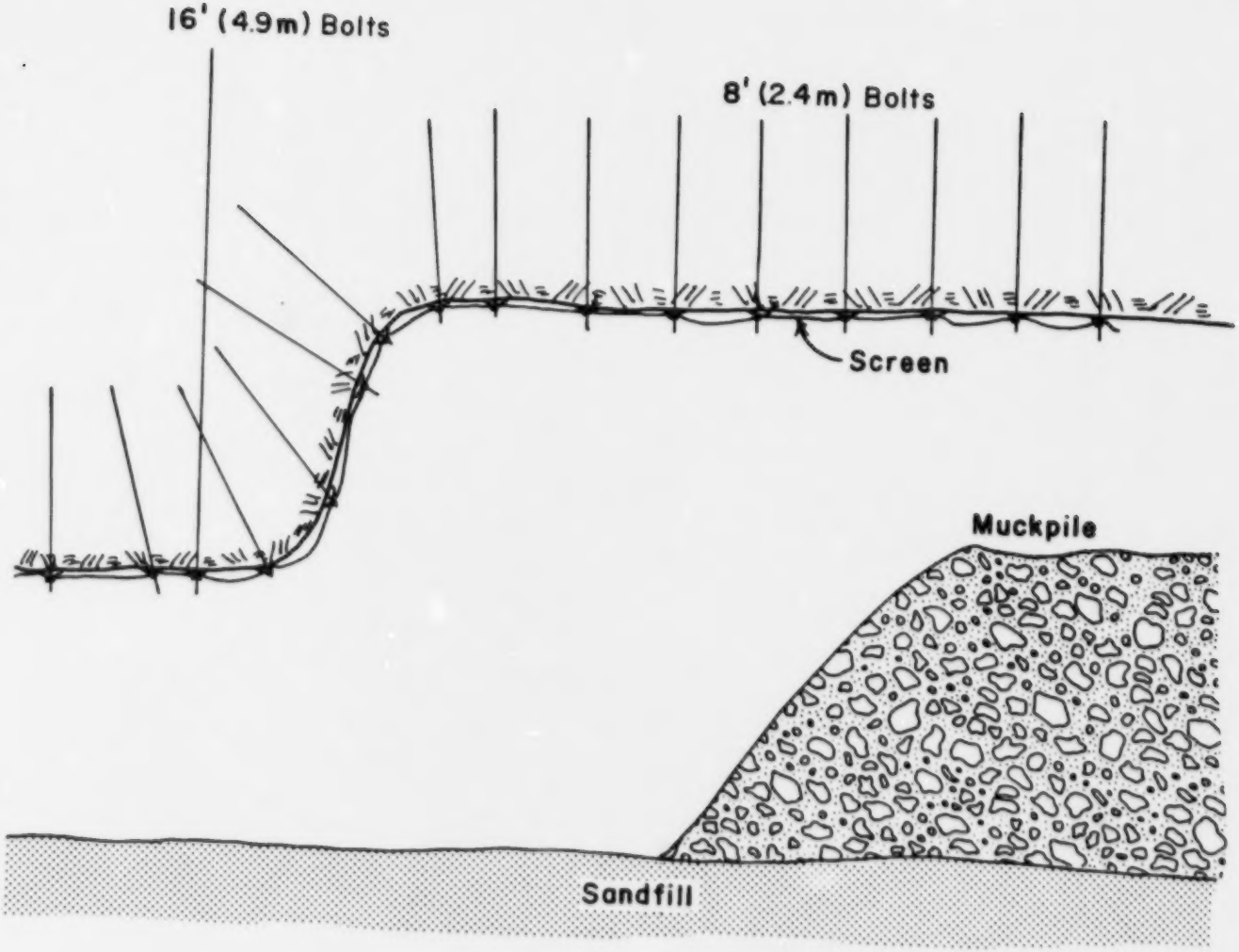
Ground Conditioning - Cross-Section Of A Typical Stope

FIGURE 1.8



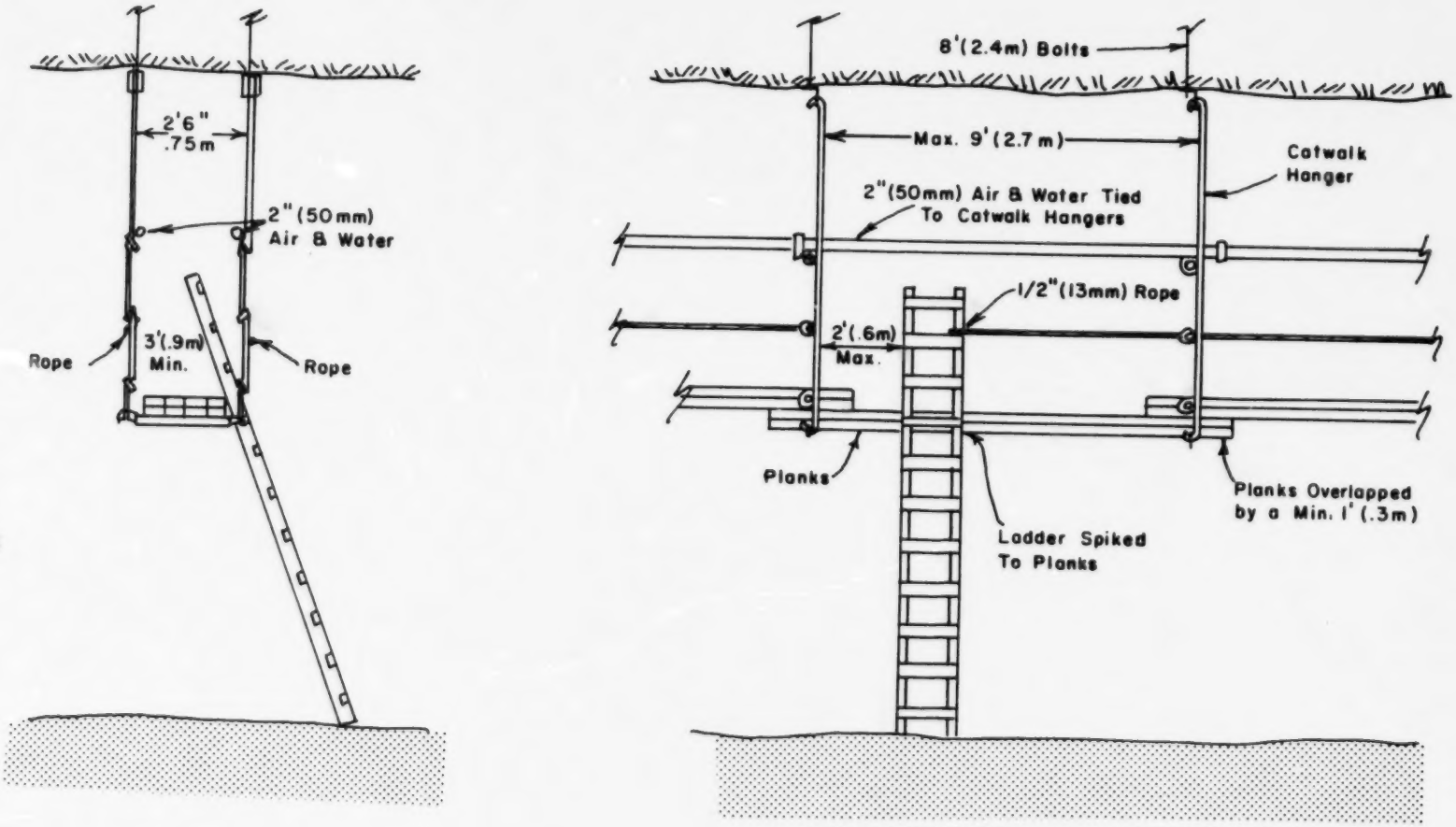
Ground Conditioning - Section Through A Typical Stope Block

FIGURE 1.9



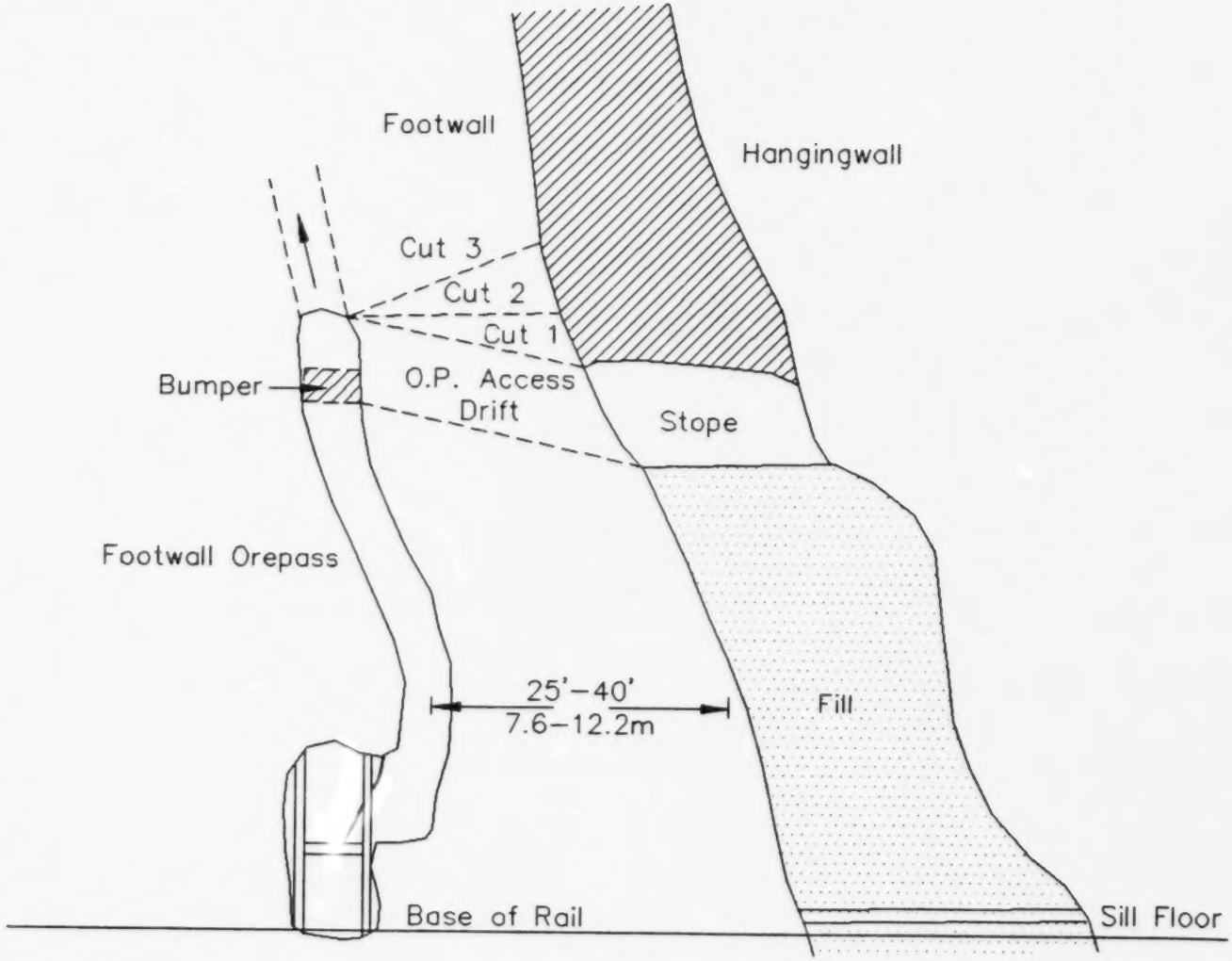
GROUND CONDITIONING - Longitudinal Of Typical Hanging Breast

FIGURE 1.10



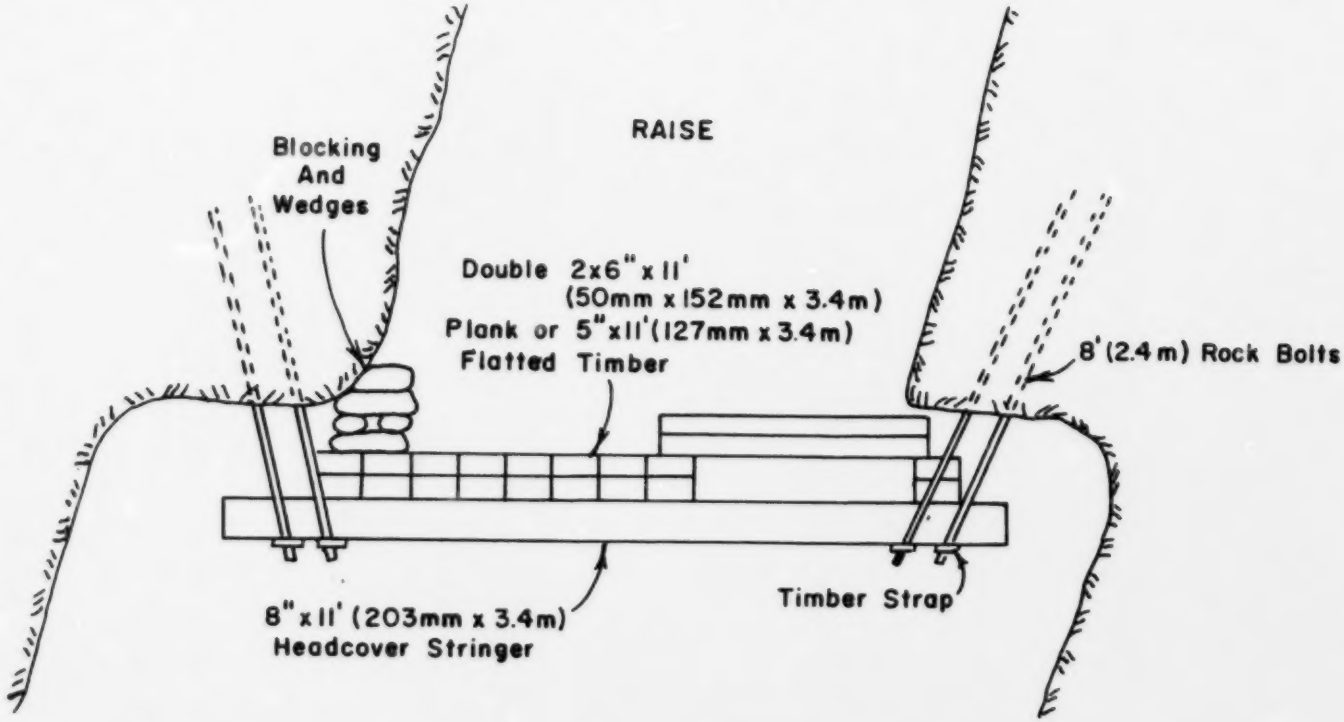
Hanging Catwalk Construction

FIGURE 1.11



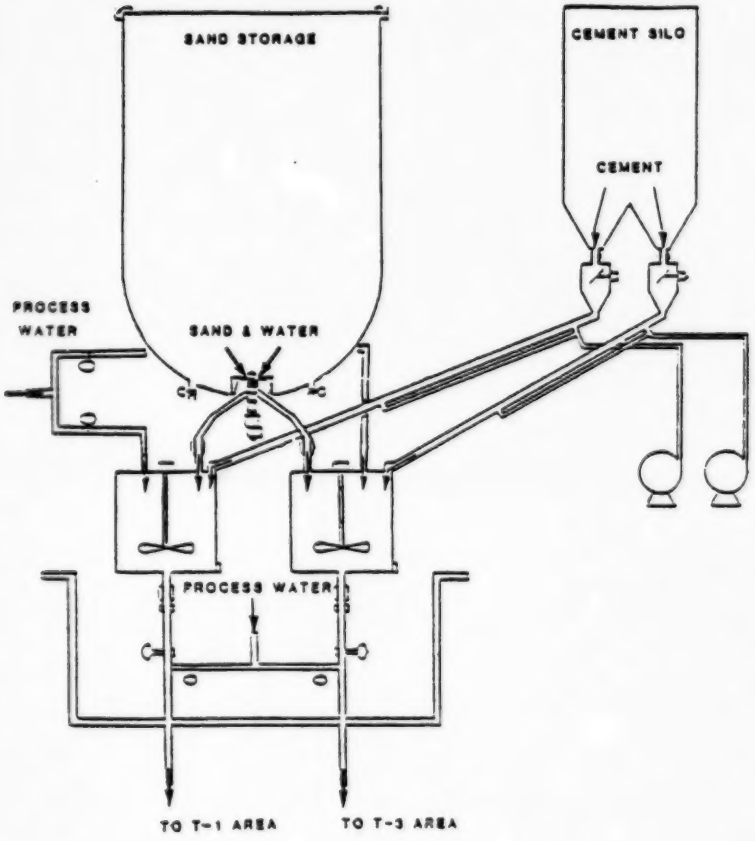
TYPICAL FOOTWALL OREPASS EXCAVATION

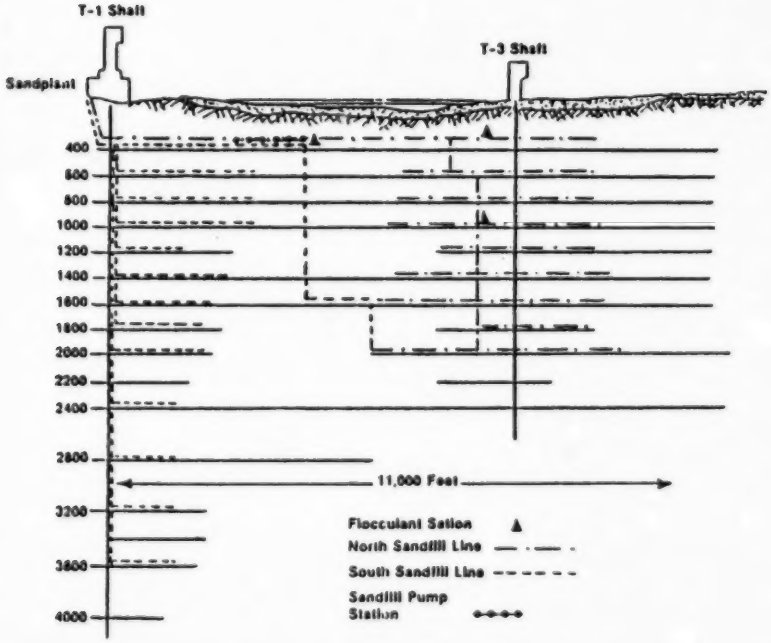
FIGURE 1.12



Standard Headcover Construction

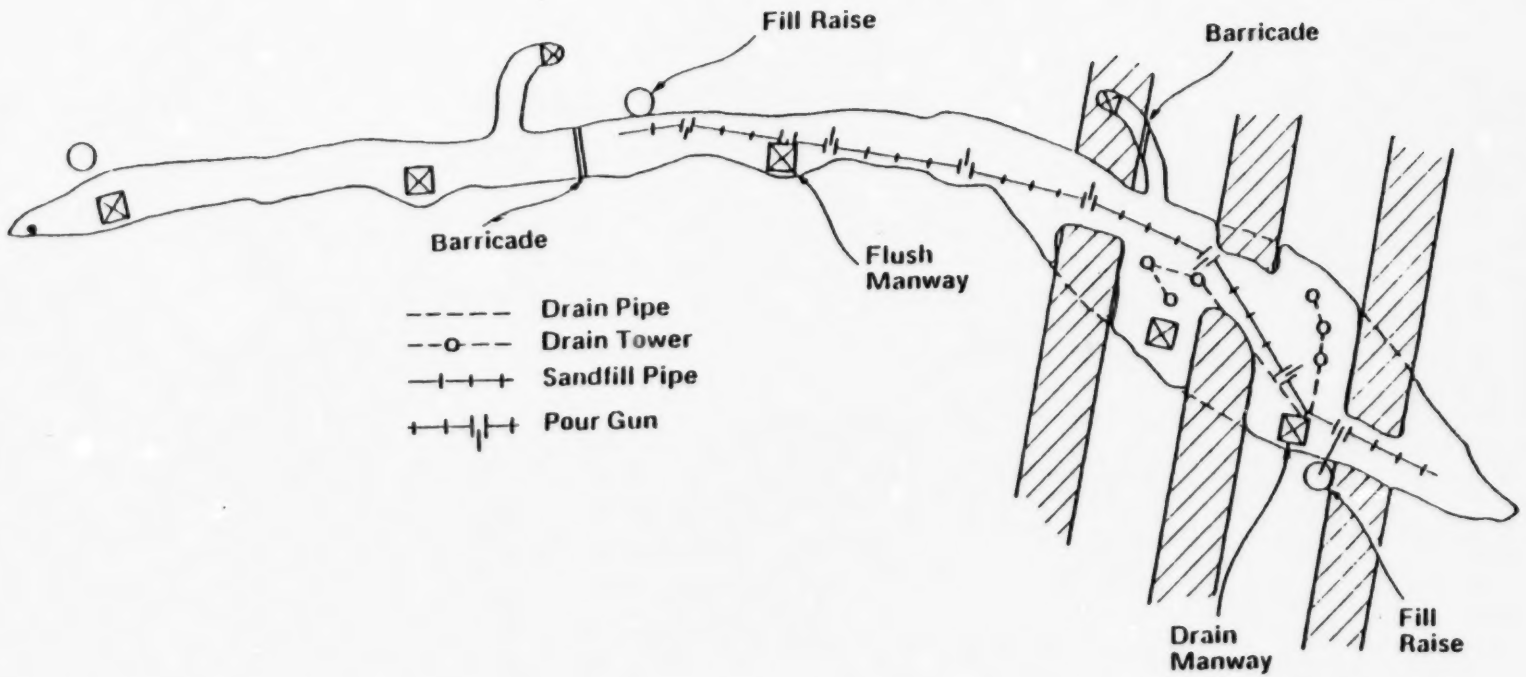
FIGURE 1.13





THOMPSON MINE SANDFILL PLANT AND DISTRIBUTION SYSTEM

FIGURE 1.14

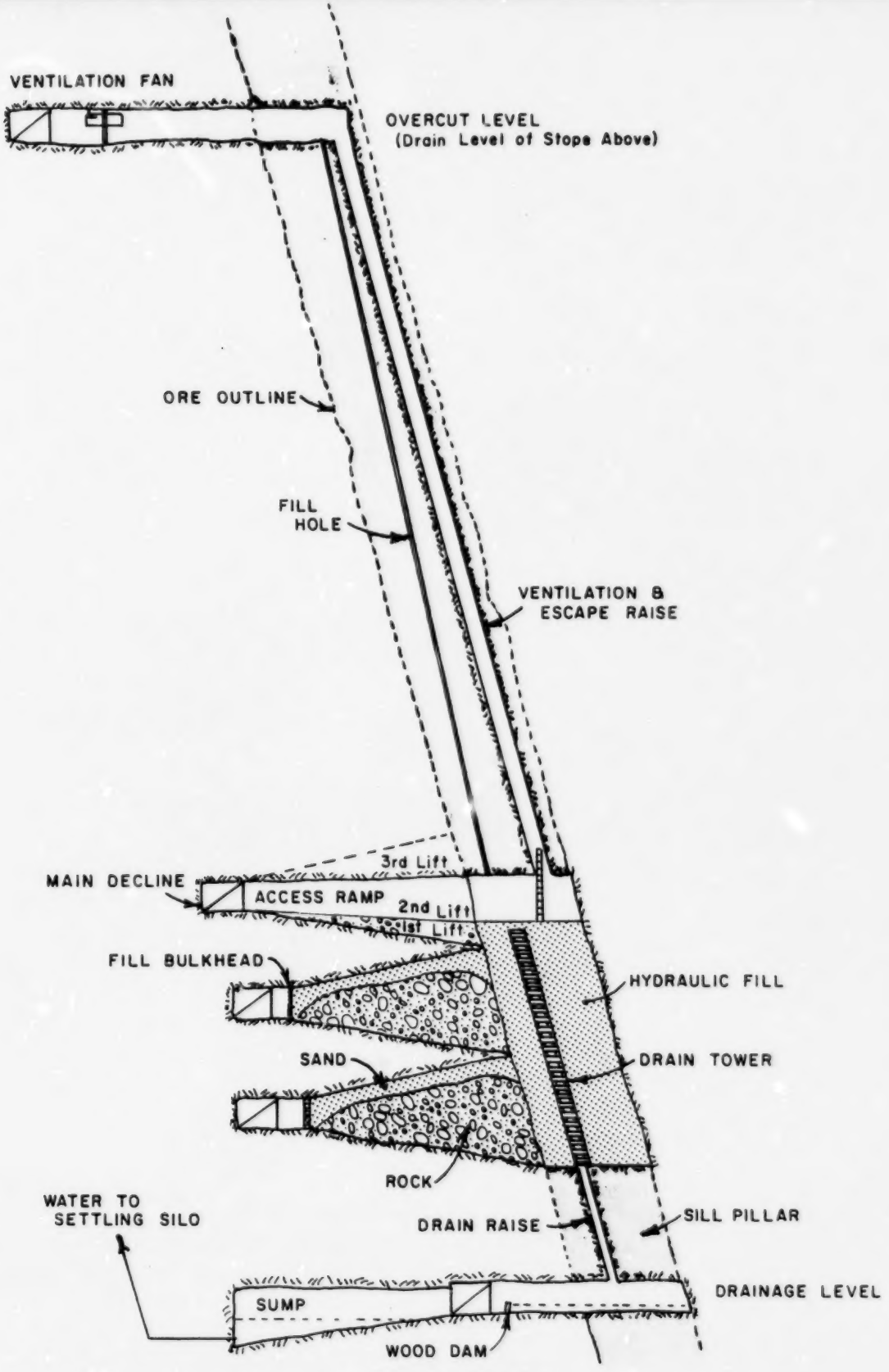


Typical Stope Prepared For Sandfill

FIGURE 1.15

TROUT LAKE OREBODY

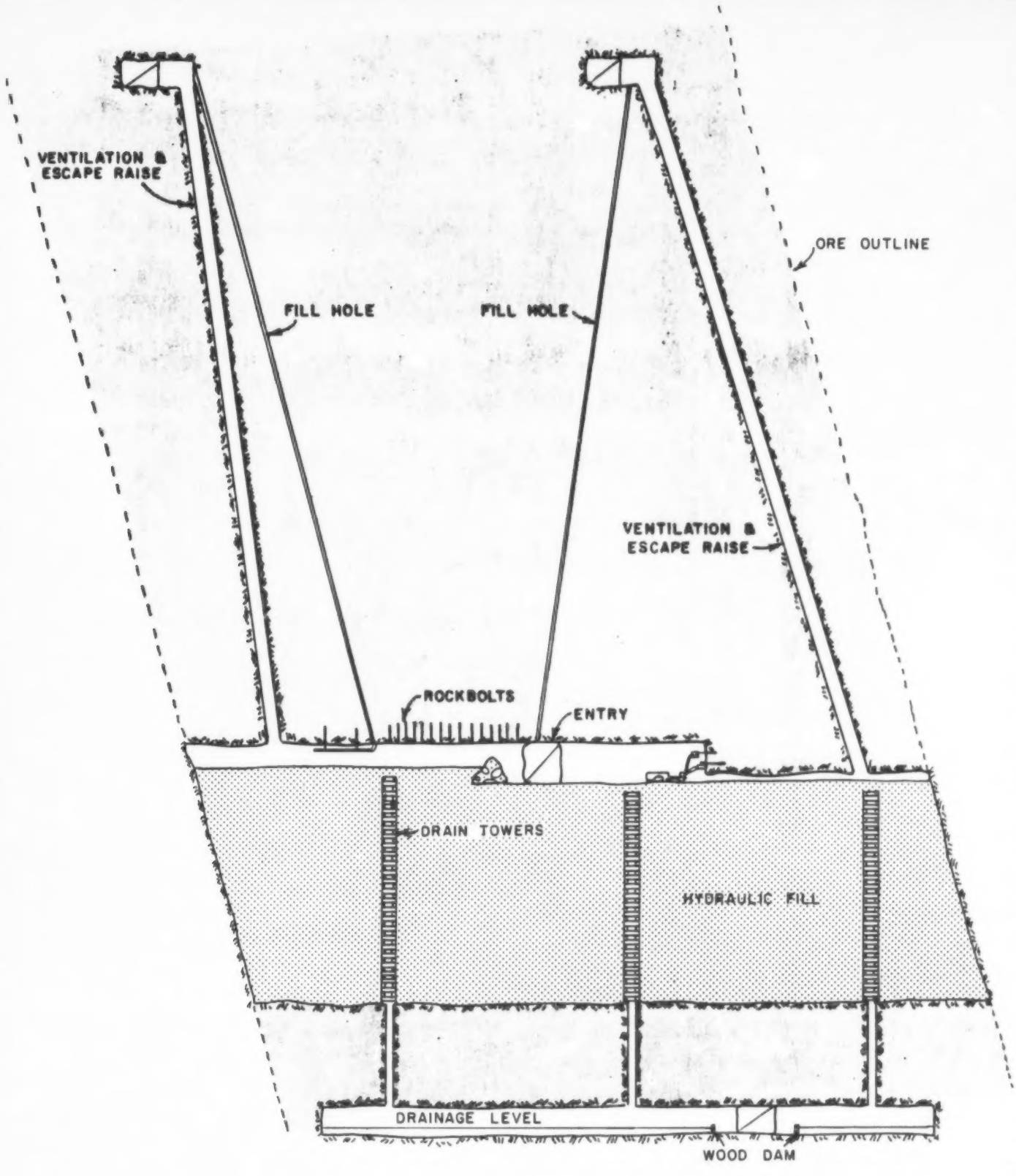
FIGURE 1.16



CROSS SECTION

TROUT LAKE MINE

FIGURE 1.17



LONGITUDINAL SECTION

TROUT LAKE MINE

FIGURE 1.18

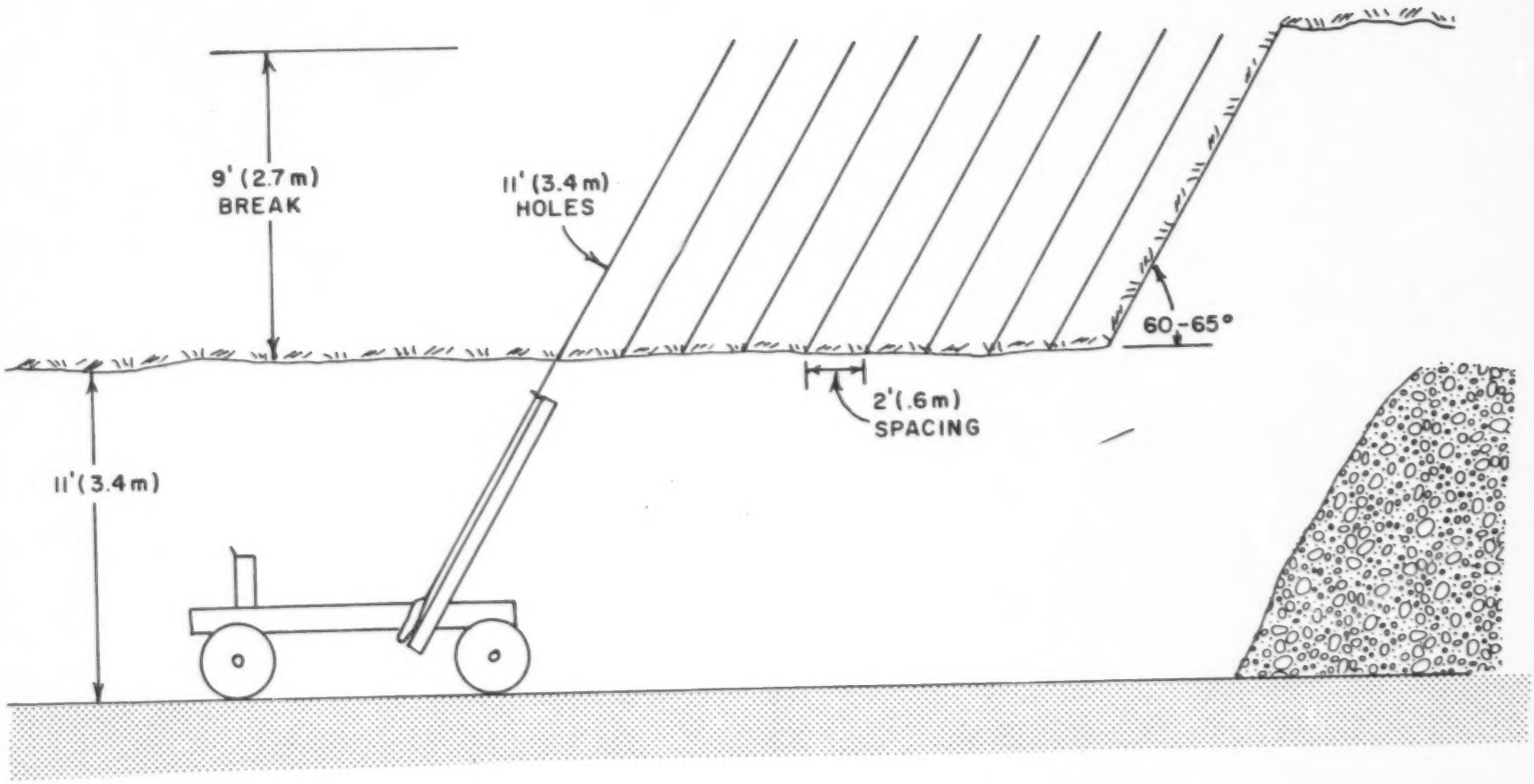
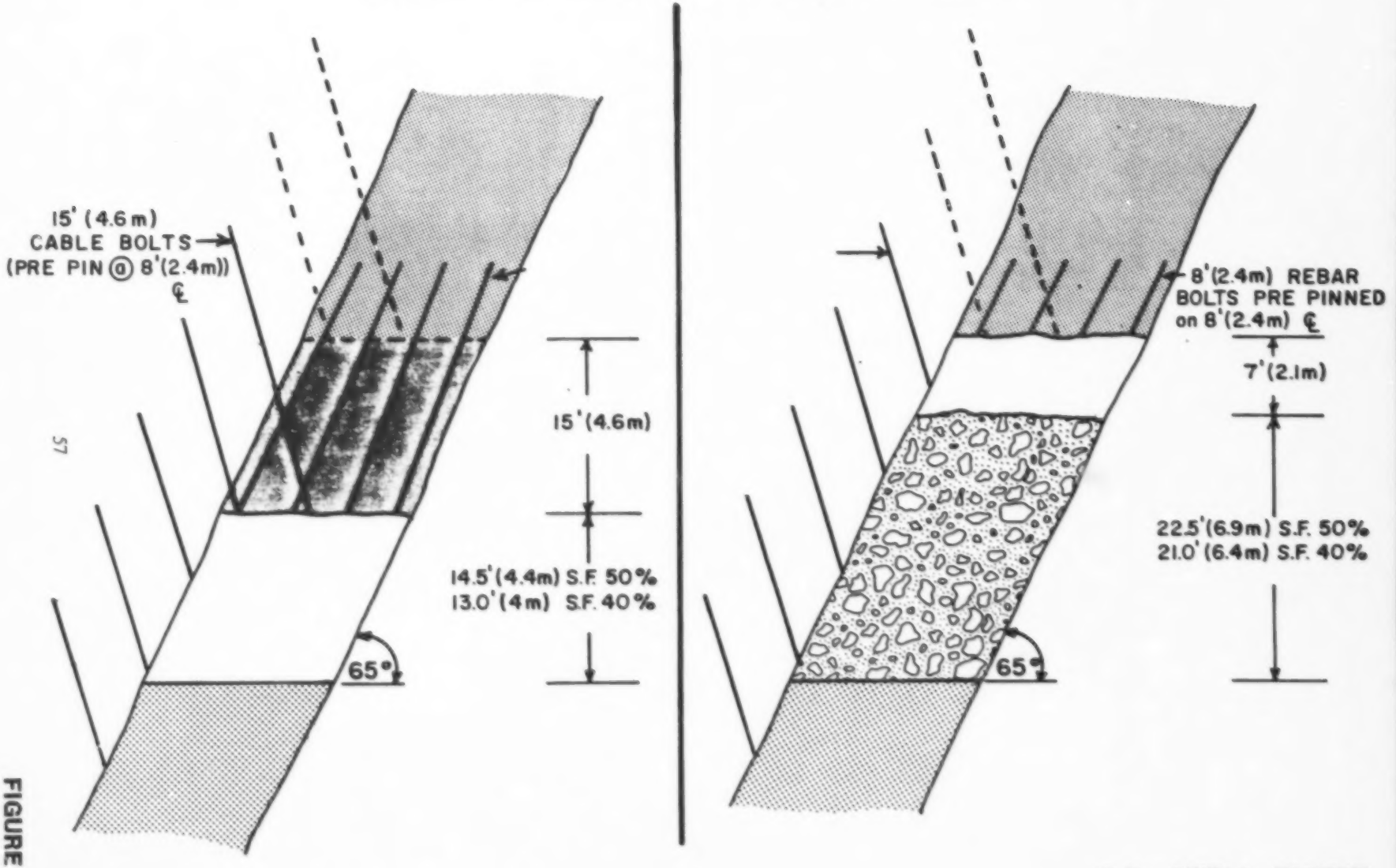


FIGURE 1.19

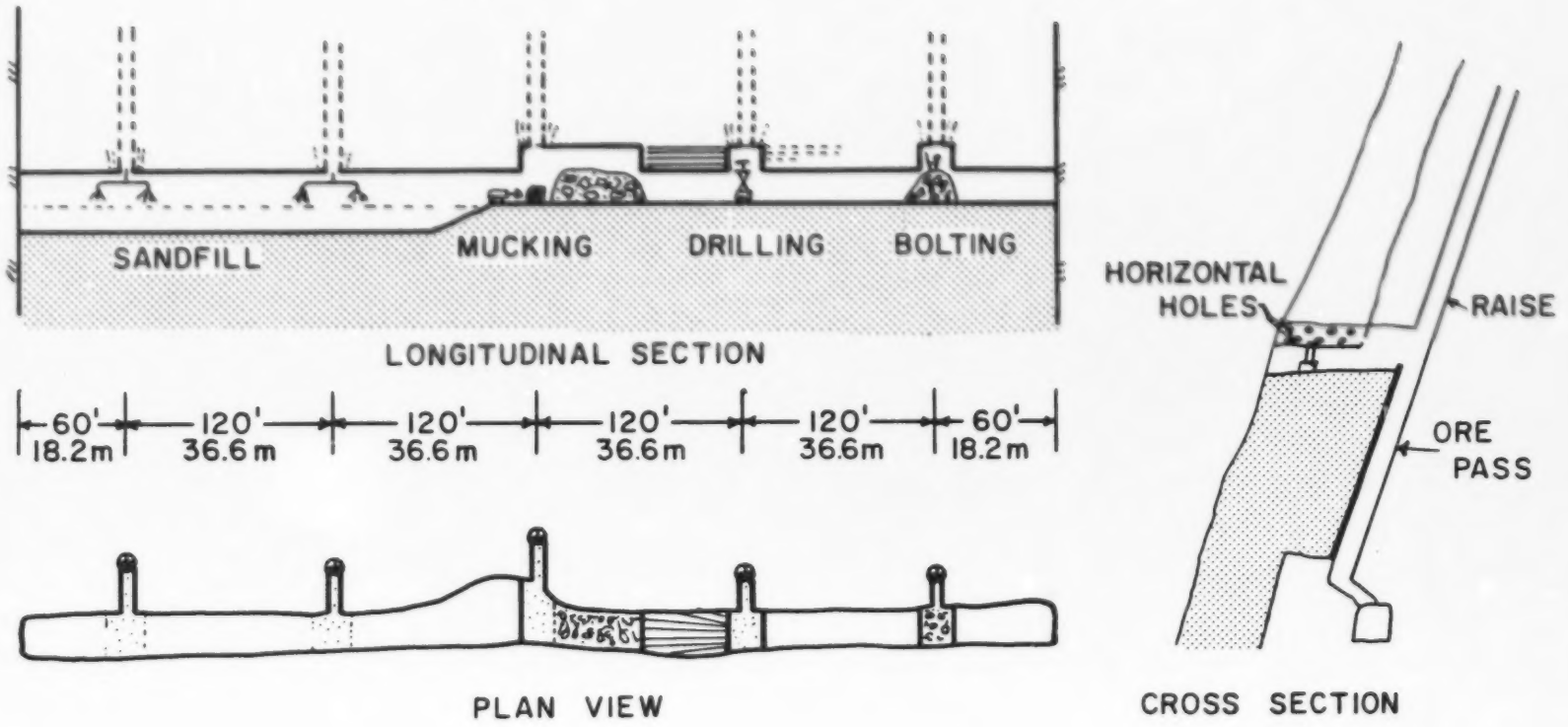
TYPICAL WAGON DRILL SET UP

PROPOSED UPPERS DRILLING



S.F. = SWELL FACTOR

FIGURE 1.20



HORIZONTAL REMOTE CONTROL MINING (IDEALIZED)

FIGURE 1.21

HORIZONTAL LONGHOLE MINING

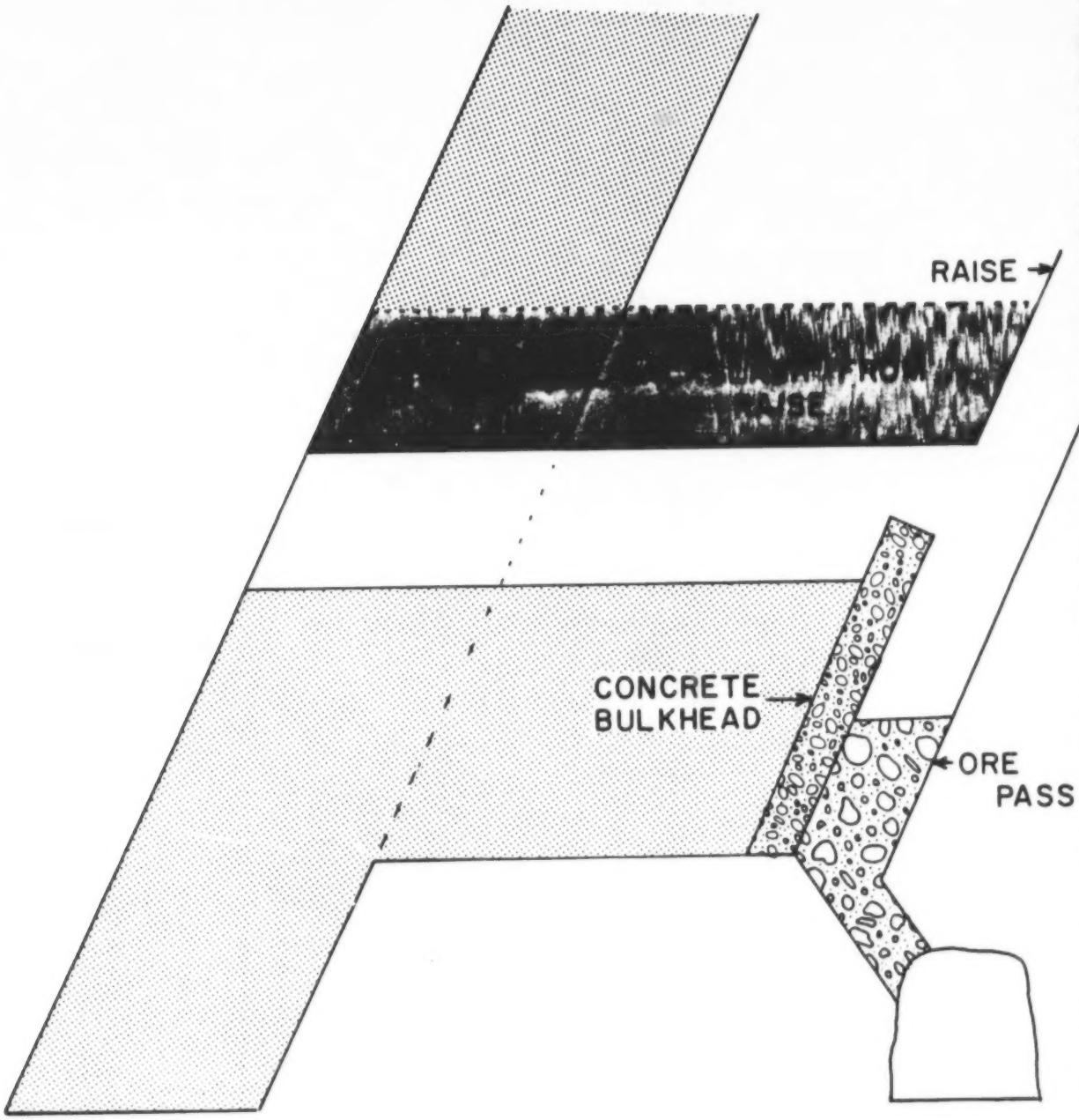


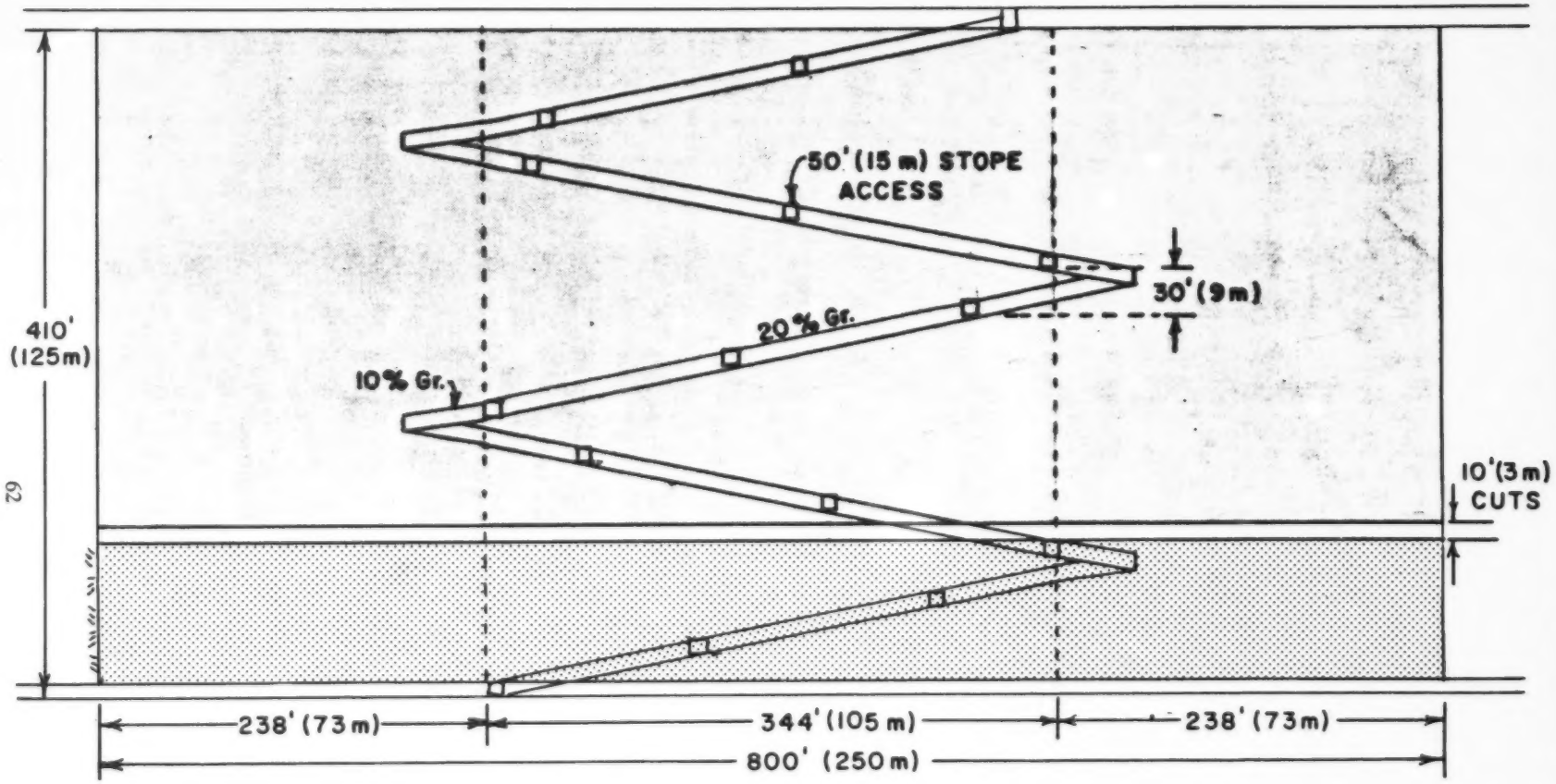
FIGURE 1.22

ALTERNATIVE "A" — RAMP IN ORE-I

FIGURE 1.23

RAMP-IN-STOPE MINING

FIGURE 1.24



TYPICAL RAMP ACCESS

FIGURE 1.25